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ACCIDENT EXPERIENCE AND DIRECT COSTS IN SOME COLORADO COAL MINES, 1929-33[1]

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This paper discusses accident experience and costs in 150 Colorado coal mines for the 5 years 1929-33; these mines produced 62 percent of Colorado's coal tonnage during this period. This and similar studies by the United States Bureau of Mines in other States (listed in the appendix to this report) make available comparable information on mine-accident costs and bring out clearly the importance of accident reduction from the humanitarian as well as from the economic standpoint.

Compensation insurance costs constitute a considerable part of the cost of coal output. The tendency toward greater liberalization of workmen's compensation laws and increased benefits thereunder may mitigate to some extent the seriousness of an accident to a worker and his dependents, but no compensation payment can restore life or limb or eliminate human suffering. Compensation is paid ultimately, not by an insurance fund but by the mine operator and the general public, which includes the miner. The one really effective method of reducing compensation costs is to reduce the number and severity of accidents.

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Summary of Compensation and Medical Costs in Colorado Coal Mines for 1929-33

1. The actual compensation cost of 2,755 compensable Colorado coal-mine accidents during the 5 years 1929-33 was $922,768.80. Estimated medical costs for these 2,755 cases, based on known medical costs for 430 cases, and estimated medical costs for 2,001 noncompensable accidents, based on known medical costs for 635 cases, aggregate $201,098.82. The direct compensation and medical cost of the 4,756 accidents (2,755 compensable and 2,001 noncompensable) was therefore about $1,124,000. These accidents occurred in mines producing about 62 percent of Colorado's coal; therefore the probable 5-year direct cost of accidents to all Colorado coal mines in compensation and medical payments was about $1,800,000. This figure does not include cost of insurance administration.

2. Each compensable case averaged $335 in compensation and $67 in medical charges.

3. The major compensation costs are for fatalities, permanent total disabilities, and permanent partial disabilities. The 8 permanent total disabilities average more than $16,000 each, fatalities more than $2,500 each, and permanent partial disabilities nearly $1,000 each.

4. Compensable injuries involving no permanent disability cost about $51 each in compensation and $46 in medical charges.

5. Noncompensable injuries cost about $8.50 each in medical charges.

6. Coal-mine accidents in Colorado cost in compensation and medical payments approximately $0.05 per ton of coal produced.

7. The causes of coal-mine accidents in Colorado were about the same as in the United States as a whole. In many instances accident reports indicate that the accidents could have been prevented. Experience in numerous mines in the United States has shown that the number and severity of accidents can be greatly decreased; in some mines accidents have been eliminated for long periods.

8. Payments to injured men during time off for accidents, exclusive of the 10-day waiting period for temporary compensable disabilities, ranged from an average of $1.63 per day in 1929 to $1.18 per day in 1933.

Important accident data for 150 Colorado coal mines, 1929-33

Number of mines	150
Coal production, 5 years ... tons	21,991,672
Man-days worked, 5 years	4,618,679
Fatal accidents	110
Permanent total disabilities	8
Temporary disabilities (compensable)	2,224
Permanent partial disabilities	413
Total compensable accidents	2,755
Total noncompensable accidents	2,001
Days lost by all accidents (compensable and noncompensable)	1,186,456
Days lost per accident (compensable and noncompensable)	249.47
Accidents per fatality	43.20
Coal produced per fatality ... tons	199,924
Coal produced per accident (compensable and noncompensable) ... do	4,624
Direct compensation costs ... cents per ton	4.2
Direct compensation and medical cost ... do	5.1
Number lost-time accidents	3,993
Frequency rate of accidents (lost-time accidents per million man-hours worked)	108.1
Severity rate (days lost per 1,000 hours worked)	32.1

NOTE. - Figures on coal production, days the mines worked, and number of men employed in each of the 150 mines, are from annual reports of the Colorado State Coal-Mine Inspector for 1929-33. Eight hours are taken as a working day.

Methods Employed in Accident Study

Through the cooperation of the Colorado Industrial Commission and the Employers Mutual Insurance Co. engineers of the United States Bureau of Mines were given access to the individual reports of coal-mine accidents occurring in mines insured by the State Fund and the Employers Mutual Insurance Co. These reports covered approximately 150 coal mines which produced approximately 22,000,000 tons of coal of a total of 35,677,618 tons for the State as a whole during the period 1929-33. Six mines operated by coal companies carrying their own insurance and small uninsured mines were not included in this study.

The Workmen's Compensation Act of Colorado (sec. 30) provides that:

> Every employer shall keep a record of all injuries, fatal or otherwise, received by his employees in the course of their employment. Within 10 days after the occurrence of an accident resulting in personal injury, a report thereof shall be made, in writing by the employer to the commission, upon forms prescribed for that purpose. Such reports shall contain such information as shall be required by the commission.

The employer's report and those made by the mine operator to the commission are in most instances fairly complete, giving information as to the place, time, and cause of the accident, length of disability, part of body injured, and employee's rate of earnings.

This paper is based on information obtained from individual files covering 4,756 compensable and noncompensable coal-mine accidents in Colorado for 1929-33, inclusive. The authors believe that it furnishes a practically complete and accurate record of the number of accidents, direct causes of accidents, nature of injuries, and compensation payments to date for all coal mines covered by compensation other than the mines of the 6 companies operated by self-insurers. Compensation payments for this 5-year period will be increased slightly by the occasional reopening of supposedly closed cases; a claim case cannot be considered absolutely closed until after the death of the claimant through natural or other causes. In a few unsettled cases of severe injuries future compensation payments have been estimated. In cases of permanent total disability a life-expectancy table (p. 54, Workmen's Compensation Act of Colorado, Expectancy Table C. L. sec. 6,537) was used in computing the future compensation payments to be made. Complete medical and hospital costs were available for tabulation and study only for claims paid by the State Fund; reports on claims of the Employers Mutual Insurance Co. covered only compensation and burial costs, the individual mine operator member taking care of medical costs. In some instances the insurance company (and the State Fund as well) assumes additional expense for extra medical, surgical, and hospital service in efforts, often successful, in lessening the degree of permanent disability of an injured person and thus enabling his return to useful employment with resultant decrease of compensation.

The number of days lost from accidents that do not involve permanent or total disability or death is the number of days elapsing from the day of the disability to the time of return to work, including Sundays and holidays. Where the injured man did not report back for work the attending doctor's estimate of time lost has been used. The compensation awards for permanent partial and total disabilities and fatalities vary in the several States, and to make figures on days lost comparable, the table (table 9) for weighting industrial accident disabilities recommended by the Association of Industrial Accident Boards and Commissions was used. Thus, a fatality or permanent total disability is rated as 6,000 days lost time, loss of a finger as 300 days, and other permanently disabling injuries proportionately. In most cases this weighting system gives a loss of time considerably greater than that figured from the award made, but in other cases involving a long period of temporary disability in addition to a permanent-partial-disability award the number of days lost by the weighting system is less.

The figures include coal-mine accidents occurring in and about the mines during 1929-33 inclusive, but do not include coal-trucking accidents. The period of disability for the compensable accidents listed was more than 10 days. In a few cases no compensation was paid because of inability to locate the injured man or failure of the injured man to make claim within the legal

6-month period, and in some fatal-accident cases only funeral costs were included because the victims had no dependents. The noncompensable cases include all the injured reported as disabled for 10 days or less; many of the no-lost-time accidents involve considerable medical cost.

Cost of Accidents

From 1929 to 1933, inclusive, there were in and about Colorado coal mines, excluding self-insurers, 2,755 lost-time accidents involving more than 10 days each and 2,001 lost-time accidents involving 10 days or less each, a total of 4,756 lost-time accidents. A total of $922,768.80 was paid in compensation (including $28,173.75 for funeral, extra surgical, hospital, dental, and miscellaneous fees) for these 2,755 compensable accidents. The medical cost of 430 compensable cases for which figures were available was $28,733.16 (66.82 each), and the reported medical cost of 635 noncompensable cases was $5,396.75 ($8.50 each). If these figures represent average medical costs of compensable and noncompensable cases the estimated total medical cost for all compensable accidents is $184,092.68 and for all noncompensable accidents, $17,006.14.

The estimated total compensation and medical fees for these 4,756 cases is therefore $1,123,867.62, of which $922,768.80 represents compensation payments.

A large part of the total compensation cost of Colorado coal-mine accidents is due to fatalities, permanent total disabilities, and permanent partial disabilities; 110 fatalities cost in compensation $277,346.46 or an average of more than $2,520 per fatality. It is estimated that 8 permanent total disabilities will cost $128,563.88 in compensation or that these men now totally blind or with broken backs or similar injuries barring them from work for life will receive, on the basis of $14 or less compensation per week, an average of more than $16,000 each.

The compensation for 413 permanent partial disabilities, many involving such serious impairment as loss of an eye, cost $403,616,84 or an average of more than $977 each. The total compensation cost of the 531 fatalities, permanent total, and partial disabilities is $809,527.18, leaving a compensation cost of $113,241.62 for the 2,224 remaining compensable injuries involving no permanent disability. The average compensable case involving no permanent disability therefore cost only $50.92 in compensation. Based on a compilation of 352 cases, such a case costs $45.78 in medical bills.

Time Lost in Accidents

The time lost by the injured in mine accidents is of interest. In the 4,756 accidents reported 1,186,456 days, an average of 249.5 days per accident, were lost; 2,755 of these accidents were compensable, and the average days lost for each compensable case was 428.2. It should be remembered that in calculating days lost a permanently disabling accident or a fatality is rated by the scale of time losses given in table 9; thus, the loss of a leg below

the knee is equivalent to 3,000 days and a fatality to 6,000 days. Of these accidents 2,001 were noncompensable, and the average time lost, including 763 no-lost-time accidents, was 3.4 days. Based on an average of 290 cases, the no-lost-time accidents averaged in medical care $7.30, and 12 of them involving no lost time but injury to teeth cost $12 to $71 each for dental care. These data are given in detail in tables 2 and 3.

Causes and Prevention of Accidents

Table 4 gives the immediate causes of accidents and average days lost from each cause according to the United States Bureau of Mines classification. However, comment on the proximate causes, based on individual accident reports, is worth while. The term "proximate" is used for the many factors that contribute to an accident; for example, condition of surroundings or equipment, system of mining, illumination, pitch of coal seam, mine practices, discipline, supervision, state of mind of the injured at time of accident, fatigue, ill health, lack of mental alertness, poor eyesight, carelessness, recklessness, loose clothing - any or several of these may be more important in causing accidents than the immediate cause ascribed.

Means of preventing mine accidents are discussed in many Bureau of Mines publications; a list of publications dealing with the prevention of any particular class of accidents may be obtained by writing to the Bureau, which has assembled general data regarding coal-mine accident prevention in illustrated lecture form and for several years has given instruction on this subject throughout the United States. The lectures require approximately 20 periods of 2 hours each.

Adequate safety supervision, discipline, and safety education of mine officials and miners are essential for the general reduction of mine accidents. Cooperation of miner and official is necessary for successful accident prevention, but management must take the lead in safety work. Proper safeguarding of mine surroundings and equipment, good upkeep, and periodic safety inspection of equipment, safety instruction, safety bulletins, and general safety meetings are means by which a company can take that leadership in safety work so necessary in obtaining the cooperation of the workers.

During the last 3 years there has been a marked reduction in the number of coal-mine accidents in the United States on a frequency as well as on a tonnage basis. Numerous mines in various sections of the United States are establishing safety records much above the average and often extending over a considerable period. More than 70 percent of the bituminous coal produced in the United States in 1932 came from mines that reported no fatal accidents. During 1934 one western coal mine produced more than 205,000 tons of coal and employed 144 men without a lost-time injury. Another western coal mine operated from April 1923 to January 1, 1935, without a fatality, producing more than 3,000,000 tons of coal and employing more than 200 men during this period. During the 1933 National Safety Competition conducted by the United States Bureau of Mines 6 bituminous coal mines working 100,000

to 200,000 man-hours operated the entire year without a lost-time accident. In at least one of these mines natural roof conditions were decidedly unfavorable.

Causes 1, 2, and 3. - Falls of rock, coal, and draw slate (see table 4) caused almost 30 percent of all accidents listed in this paper. The average number of days lost per injury from roof falls (416) was about 60 percent more than from the average injury. Falls of roof or coal (table 7) caused 59 (about 54 percent) of the 110 fatalities in these mines. In Colorado and in the United States as a whole about half of all bituminous coal-mine fatalities and 30 percent of all nonfatal injuries are caused by falls of rock, coal, or draw slate or falls of face or rib. Briefly, essentials in reducing accidents from roof falls, particularly at the working face, are frequent and careful testing of roof by miners and officials to determine its safety, the immediate barring down or supporting of unsafe roof, and an adequate system of timbering suited to the particular mine. An adequate timbering system should provide not only for normal timbering requirements in a working place but also for additional timbering under roof conditions that are below the average; timbers must be set where and when needed. As long as the miner determines the need and position of a prop the toll of lives of both experienced and inexperienced miners will continue; on the other hand, systematic timbering, supervision, and discipline unquestionably can prevent roof-fall accidents and in many instances have greatly reduced the accident total. Roof testing by the vibration method where practicable should disclose any weakness in large slabs found not drummy by ordinary striking. Even where timbering is done systematically in some mines an occasional so-called kettle bottom or pot of rock will fall; if such abnormal roof defects are frequent the use of lagging at least should give warning so that the men could escape. Mining methods and speed of extraction are also vitally important in their relation to roof-fall accidents; poor mining methods are responsible for many mine accidents.

As a preventive of fatal and serious injuries from falls of smaller amounts of rock or coal the protective hat is excellent, but it should not be expected to take the place of adequate timbering. In many Colorado mines protective headgear is required underground and about the tipple. Many lives have been saved and serious injuries prevented by wearing protective hats in and around mines.

Cause 6. - Falls of persons underground caused about 3.7 percent of all accidents. Many of these accidents were due to stumbling. Elimination of stumbling hazards requires that travelways be kept as clear as possible and adequate illumination be provided where the hazard cannot be avoided readily.

Cause 7. - Handling materials caused about 11 percent of all accidents. Such accidents are less severe than the average. Protection of hands and feet is of considerable importance in preventing them; suitable gloves would reduce hand injuries, and protective shoes would decrease foot injuries.

Causes 8 and 23. - Hand tools caused 16 percent of all injuries; most of these injuries were relatively slight. The use of proper tools is important in avoiding injuries; sharp picks and axes and good handles tend to decrease accidents from tools. Numerous eye injuries are included in accidents from hand tools. Sometimes the report was not specific as to the cause of the eye injury, the accident being reported as due to flying coal or rock; possibly some of these injuries were received while shoveling coal or rock, but most of them were received during picking and hammering.

Of the 4,756 accidents reported (tables 4 and 5), 531 involved eye injuries; many of these were trivial and caused no lost time, but in 65 cases the result was total or partial loss of one eye. The average compensation cost was more than $700 and the average number of days lost, more than 1,100 each. Protective goggles are supplied the miners in many Colorado mines, and their use is required in certain mining operations, particularly picking. Nevertheless, it is manifest that their actual use is by no means universal, as indicated by the 120 eye injuries received in 1933.

Cause 9. - Stepping on nails or other sharp objects caused relatively few injuries with little cost and time lost. Good mine housekeeping involving general orderliness and removal of protruding nails from boards, ties, etc., is important in eliminating such injuries, as are good soles on shoes.

Cause 11. - In Colorado, as in other States, mine haulage ranks next to roof falls as a cause of mine accidents. Haulage accidents also have practically the same high severity rate as accidents from roof falls. During the period 1929-33 20.38 percent of all mine accidents in Colorado occurred in underground haulage and about 1 percent in surface haulage. In more than half of these accidents the men were struck, run over, or squeezed between cars or between car and rib, timber, or roof. Relatively few men are employed in haulage compared with other mine operations, and the ratio of fatal and nonfatal accidents to the number of men employed, is high in Colorado and in the United States as a whole. Supervision of haulage work is always difficult. Adoption of definite rules regarding haulage practices, insistance on compliance therewith, and careful selection of haulage employees would reduce such accidents. Some of the other factors entering into haulage-accident prevention are proper equipment, proper maintenance of equipment and haulageways, suitable lay-out of haulage for safe as well as efficient haulage of coal, and maintenance of adequate man clearance along haulageways. The human equation undoubtedly is important in the occurrence as well as in the prevention of haulage accidents in coal mining, but the responsibility of the management as regards haulage equipment, installations, and methods is much greater in connection with haulage accidents than is usually believed.

Cause 12. - Before 1928 numerous gas and dust explosions occurred in Colorado bituminous-coal mines with great loss of life and damage to property. During recent years, however, there have been few explosions from this cause in Colorado mines. Factors contributing to this improvement are probably better mine ventilation, better company inspection of mines and testing for gas, vigilance on the part of the State mine-inspection department, and

probably several others; unquestionably another contributory element is the matter of good fortune or luck. During 1929-33, 11 men were burned in Colorado coal mines by gas, indicating that the old danger of ignition of explosive gas still exists and that gas is still being ignited occasionally with consequent danger of a general dust explosion and disaster. Rock-dusting as a means of preventing the spread of coaldust explosions in coal mines generally is known but is adequately practiced in only a few Colorado coal mines.

Cause 13. - Explosives caused relatively few accidents (24 or 0.05 percent of all accidents), but such injuries were usually serious, the severity rate being about twice that of the usual type of mine accident. In many Colorado mines accidents from explosives probably have been prevented by the practice of blasting by shot firers after the shift is out of the mine. The present tendency in Colorado is toward increased use of explosives during the working shift, particularly where mechanical loading has been introduced; the shots are fired by miners as well as by shot firers. This tendency is likely to increase the number of explosives accidents as well as mine explosions; only by careful supervision of explosives practice can serious accidents be avoided when blasting is done during the working shift.

Cause 14. - Electricity caused relatively few accidents, but the severity rate and compensation cost were high owing to 7 electrocutions from contact with trolley and power lines and electrical equipment during the 5-year period. Adequate guarding of trolley and power lines and proper grounding of fixed and portable electrical equipment are factors in eliminating such accidents. As long as men are exposed to long stretches of unguarded trolley and power lines 6 feet or less above the rail, electrocutions will be numerous. Experience of electric-power companies has amply demonstrated the danger to workers of even 110-volt current; voltages of 220 to 500 in the mines are correspondingly dangerous.

The electric contact hazard in mines is intensified because the electric wiring and equipment must be installed in a relatively confined space, usually near places where workmen must pass or labor, the underground illumination is generally poor compared with that on the surface, and the men may contact rails, pipes, and water at the same time they contact a power conductor. Elimination of this hazard appears to be mainly a problem of management.

Cause 17. - Two fatalities caused by escape of gases from a sealed mine fire illustrate the danger of inadequately sealed and inspected fire areas.

Accidents Classified According to Parts of Body Injured

Table 5 classifies accidents according to parts of the body injured; the accidents listed under head, eye, hand and finger, and foot involve injury only to one part of the body. Accidents that result in injuries to several parts of the body are included under miscellaneous injuries with all other accidents not falling into the first four classifications. Table 5 further classifies accidents according to number involving total or partial loss of eye and number of hernia injuries. The number of head injuries (157) is relatively small, possibly due in part to extensive use of protective hats,

although accidents with head injuries do not appear to be decreasing in number. In many instances of severe injury to the head, as from the falling of a large slab of rock, other parts of the body are injured also; these accidents appear under "miscellaneous." Eye injuries have been discussed briefly under accidents from hand tools; 531 eye injuries occurred during the 5-year period - 120 in 1933, more than in any of the 4 preceding years. During this year, however, there was a material reduction in the number of cases of total or partial loss of eyesight with 7 such cases against 19 in 1929. Experience of some mining companies indicates that virtually all eye injuries could be eliminated by the use of goggles. Objections to the use of goggles are: The wire-screen type reduces illumination approximately 20 percent; the glass goggle, particularly in warm and poorly ventilated places, tends to fog and wiping the glass continually smears it with dirt; a wire-screen goggle or an uncorrected glass goggle may hinder a man with poor eyesight and increase his liability to other accidents, as from roof falls or haulage. Nevertheless, many companies have found that these apparent difficulties can be overcome and that by rigid supervision the men can be induced to wear goggles during underground work. The safety engineer of a large western company employing more than 1,000 men underground stated that eye injuries formerly costing up to several thousand dollars a year had been eliminated by examination of the eyes and compulsory use of goggles with lenses fitted to correct defects in vision; approximately 1½ years after their introduction there were no eye injuries in the mines of this company. The men are required to use goggles during all underground mining operations. Another western company which began to use goggles 6 or 7 years ago also has virtually eliminated eye injuries.

More than 1,000 hand and finger injuries were reported. Most of these were slight, but there were numerous instances of loss of one or more fingers or parts thereof.

Foot injuries (765) also show a comparatively low severity rate, although in the 5-year period they have not tended to decrease in number. More general use of protective footwear should, however, decrease such accidents both in frequency and in severity.

During the period studied hernia cases increased from 12 in 1929 to 23 in 1933 in spite of the fact that the law in Colorado provides that (1) the appearance of hernia must have been accompanied by pain and (2) that it was immediately preceded by some accidental strain suffered in the course of employment. The law also provides that an employee may elect to be operated upon; in the event that he does not so elect he loses the right to compensation for further aggravation of the injury.

Appendix

The tables (1 to 9) in this appendix give detailed information on the cost, frequency, severity, and causes of accidents in the majority of Colorado coal mines for 1929-33, inclusive. Information also is included regarding the part of the body injured and the average daily payments to injured men. The main facts disclosed by these tables have been discussed in the body of this paper.

Table 1 gives by years the compensable accidents in approximately 150 Colorado mines. The number is approximate because some mines closed and others began operations during the 5-year period studied. The days lost from injury are actual days lost for nonpermanent disabling accidents but are weighted losses for fatalities and permanent disabilities, as shown in table 9. The compensation costs cover actual payments made to injured persons or their dependents and extra medical care not provided by law but do not include the cost of insurance administration. The medical costs are for the cases for which medical cost was reported, about one sixth of all compensable cases; from this partial medical-cost survey an estimate has been made of total medical cost for all cases.

Table 2 gives by years the noncompensable accidents in the same mines. Medical costs were reported for about one third of the cases, and from these the total medical cost of all noncompensable accidents was estimated. As shown in the table, several of these cases involved no loss of time but did involve considerable medical cost.

Table 3 is a composite summary for the 5-year period of compensable and noncompensable accidents classified separately according to cause. The average number of days lost and average compensation paid for each cause shown in this table indicate the relative accident severity rate for each cause.

Table 4 is a composite summary for the 5-year period of all accidents by causes and shows days lost, percentage of accidents from each cause, and average number of days lost from each cause. The relative frequency and severity of accidents from each cause are included.

Table 5 classifies accidents in Colorado coal mines by years according to parts of the body injured and lists separately cases of loss of vision and hernia. It also gives the cost and time losses for these various classes of injuries.

Table 6 gives the cost by year and average cost of each case of fatal and permanently disabling accidents and table 7 the causes of the 110 fatal accidents that occurred during the 5-year period.

Table 8 gives the average daily compensation paid per case for compensable temporary disabilities; this has decreased yearly owing to the annually reduced working time.

Table 9 shows the scale of time losses used in this paper for weighting permanent partial and total disabilities and fatalities.

TABLE 1. - Compensable accidents in 150 Colorado coal mines, 1929-33

Year	Compensable accidents	Days lost	Compensation cost, all accidents	Medical cases reported	Medical cost of reported cases	Estimated total medical cost	Estimated total compensation and medical cost
1929	621	361,294	$281,743.20	79	$5,653.95		
1930	507	269,429	215,878.16	60	4,618.09		
1931	558	172,071	152,385.11	88	6,324.85		
1932	538	215,429	150,447.98	94	5,735.77		
1933	531	161,357	122,314.35	109	6,400.50		
Total	2,755	1,179,580	$922,768.80	430	$28,733.16	3 $184,092.68	$1,106,861.48
Average	1 551	1 235,916	334.94		2 66.82		2 401.76

1 Per year.
2 Per case.
3 Actual medical costs were available for 430 of the 2,755 cases reported; from these costs the estimated total medical cost was calculated.

TABLE 2. - Noncompensable accidents in 150 Colorado coal mines, 1929-33

Year	All noncompensable accidents	Days lost	Medical cases reported	Medical cost of reported cases	Estimated total medical cost	No-lost-time accidents	Medical cases reported	Medical cost of reported cases	Estimated total medical cost
1929	340	1,469	68.	$630.60		93	28	$187.60	
1930	343	1,157	77	612.45		139	32	199.00	
1931	416	1,346	189	1,520.60		165	83	569.10	
1932	444	1,357	155	1,257.50		135	83	591.70	
1933	458	1,547	146	1,375.60		181	64	568.50	
Total	2,001	6,876	635	$5,396.75	3 $17,006.14	763	290	$2,115.90	4 $5,567.01
Average	1 400	1 1,375		2 8.50		1 153		2 7.30	

1 Per year.
2 Per case.
3 Actual medical costs were available for 635 of the 2,001 cases reported; from these costs the estimated total medical cost was calculated.
4 Actual medical costs were available for 290 of the 763 cases reported; from these costs the estimated total medical cost was calculated.

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TABLE 3. - Accident causes, costs, and days lost in 150 Colorado coal mines, 1929-33

Cause	Compensable accidents: No. of cases	Days lost	Average days lost	Compensation cost	Average compensation cost	Noncompensable accidents: No. of cases	Days lost	Average days lost
Underground								
1. Falls of roof (rock, coal, or draw slate)	867	531,767	613.34	$427,956.85	$493.61	414	1,580	3.82
2. Falls of roof due to car or machine knocking out post	1	6,000	6,000.00	4,419.45	4,419.45			
3. Falls of face or rib	87	26,309	302.40	17,948.45	206.30	42	111	2.64
4. Rush of coal rock or gob	8	632	79.00	711.72	88.96	3	14	4.67
5. Other falling material or objects (not being handled by injured)	40	6,247	156.18	5,098.28	127.46	36	162	4.50
6. Falls of persons	85	29,583	348.04	22,318.46	262.57	90	330	3.67
7. Handling material -								
A. Coal or rock	219	29,523	134.81	31,287.59	142.87	159	547	3.44
B. All other material	53	6,791	128.13	7,425.95	140.11	96	329	3.43
8. Hand tools	267	58,420	218.80	42,699.79	159.92	421	1,434	3.41
9. Stepping on nails or other sharp objects	3	48	16.00	25.72	8.57	13	34	2.62
10. Striking or bumping against objects	19	570	30.00	450.39	23.70	23	70	3.04
11. Mine cars and mine locomotives -								
A. Struck, run over or squeezed between	258	162,541	630.00	157,894.45	611.99	96	386	4.02
B. Squeezed between car and rib, timber and roof	184	69,843	379.58	50,125.24	272.42	87	343	3.94
C. Derailments	34	14,953	439.79	8,935.70	262.81	8	39	4.88
D. Rerailing cars	53	5,171	97.57	7,072.98	133.45	11	22	2.00
E. Pushing or pulling cars by hand (strain from)	26	1,758	67.62	4,633.69	178.22	19	82	4.32

TABLE 3. - Accident causes, costs, and days lost in 150 Colorado coal mines, 1929-33 - Continued

Cause	Compensable accidents					Noncompensable accidents		
	No. of cases	Days lost	Average days lost	Compensation cost	Average compensation cost	No. of cases	Days lost	Average days lost
Underground - con.								
F. Animals on haulage	66	19,152	290.18	$14,428.85	$218.62	50	215	4.30
G. Falling from cars (not run over)	9	6,347	705.22	2,089.01	232.11	6	20	3.33
H. Other haulage accidents	38	4,253	111.92	3,041.03	80.03	24	80	3.33
12. Explosions of gas or coal dust	10	6,363	636.30	4,889.81	488.98	1	9	9.00
13. Explosives not including explosions from gas or dust	17	12,469	733.47	5,362.28	315.43	7	51	7.29
14. Electricity (not resulting in explosions)	18	52,210	2,900.56	18,281.89	1,015.66	13	43	3.31
15. Machinery (not including 12 and 14)								
A. Mining machines	133	19,011	142.94	19,424.58	146.05	60	202	3.37
B. Drills	13	3,350	257.69	1,646.61	126.66	9	15	1.67
C. All other	22	8,267	375.77	2,980.33	135.47	9	34	3.78
16. Suffocation from mine natural gases (not from fires or explosions)						1	10	10.00
17. Mine fires (burns, suffocations, etc.)	2	12,000	6,000.00	4,278.13	2,139.06	3	0	0
18. All other accidents underground	17	601	35.35	538.20	31.66	50	96	1.92
Total underground	2,549	1,094,179	429.26	$865,965.43	$339.73	1,751	6,258	3.57

TABLE 3. - Accident causes, costs, and days lost in 150 Colorado coal mines, 1929-33 - Concluded

Cause	Compensable accidents					Noncompensable accidents		
	No. of cases	Days lost	Average days lost	Compensation cost	Average compensation cost	No. of cases	Days lost	Average days lost
Shaft								
19. Vertical or inclined shaft	15	8,564	570.93	$5,867.26	$391.15	6	7	1.17
Surface								
20. Mine cars or locomotives	20	2,253	112.65	1,274.29	63.71	18	55	3.06
21. Railroad cars and locomotives	8	12,956	1,619.50	4,444.36	555.54	7	34	4.86
22. Handling materials	21	2,123	101.10	905.07	43.10	52	95	1.83
23. Hand tools	26	3,916	150.62	1,938.30	74.55	34	83	2.44
24. Falls of persons	45	18,261	405.80	15,512.35	344.72	35	110	3.14
25. Falling objects	13	1,983	152.54	1,564.56	120.35	20	39	1.95
26. Machinery	31	24,021	774.87	15,160.88	489.06	25	45	1.80
27. Electricity	1	13	13.00	6.00	6.00	7	21	3.00
28. Explosives	1	27	27.00	12.13	12.13	9	33	3.67
29. All other surface accidents	24	11,156	464.83	9,987.78	416.16	37	96	2.59
Total surface	190	76,709	403.73	$50,805.80	$267.40	244	611	2.50
Cause unknown	1	128	128.00	$130.31	$130.31			
Grand total	2,755	1,179,580	428.16	$922,768.80	$334.94	2,001	6,876	3.44

TABLE 4. - Colorado compensable and noncompensable coal-mine accidents by cause, percentage, and days lost, 1929-33

Cause	No. of cases	Percent of total	Days lost	Average days lost
Underground				
1. Falls of roof (rock, coal, or draw slate)	1,281	26.94	533,347	416.35
2. Falls of roof due to car or machine knocking out post	1	.02	6,000	6,000.00
3. Falls of face or rib	129	2.71	26,420	204.81
4. Rush of coal, rock, or gob	11	.23	646	58.73
5. Other falling material or objects (not being handled by injured)	76	1.60	6,409	84.33
6. Falls of persons	175	3.68	29,913	170.93
7. Handling materials				
A. Coal or rock	378	7.95	30,070	79.55
B. All other material	149	3.13	7,120	47.79
8. Hand tools	688	14.47	59,854	87.00
9. Stepping on nails or other sharp objects	16	.34	82	5.12
10. Striking or bumping against objects	42	.88	640	15.24
11. Mine cars and mine locomotives				
A. Struck, run over, or squeezed between	354	7.44	162,927	460.25
B. Squeezed between car and rib, timber, or roof	271	5.70	70,186	258.99
C. Derailments	42	.88	14,992	356.95
D. Rerailing cars	64	1.35	5,193	81.14
E. Pushing or pulling cars by hand (strain from)	45	.95	1,840	40.89
F. Animals on haulage	116	2.44	19,367	166.96
G. Falling from cars (not run over)	15	.32	6,367	424.47
H. Other haulage accidents	62	1.30	4,333	69.89
12. Explosions of gas or coal dust	11	.23	6,372	579.27
13. Explosives (not including explosions of gas or dust)	24	.50	12,520	521.67
14. Electricity (not resulting in explosions)	31	.65	52,253	1,685.58
15. Machinery (not including 12 and 14)				
A. Mining machines	193	4.06	19,213	99.55
B. Drills	22	.46	3,365	152.95
C. All other	31	.65	8,301	267.77
16. Suffocation from mine gases (not from fires or explosions)	1	.02	10	10.00
17. Mine fires (burns, suffocations, etc.)	5	.11	12,000	2,400.00
18. All other accidents underground	67	1.41	697	10.40
Total underground	4,300	90.42	1,100,437	255.92

TABLE 4. - Colorado compensable and noncompensable coal-mine accidents by cause, percentage, and days lost, 1929-33 - Continued.

Cause	No. of cases	Percent of total	Days lost	Average days lost
Shaft				
19. Vertical or inclined shaft	21	.44	8,571	408.14
Surface				
20. Mine cars or mine locomotives	38	.80	2,308	60.74
21. Railway cars and locomotives	15	.32	12,990	866.00
22. Handling materials	73	1.53	2,218	30.38
23. Hand tools	60	1.26	3,999	66.65
24. Falls of persons	80	1.68	18,371	229.64
25. Falling objects	33	.69	2,022	61.27
26. Machinery	56	1.18	24,066	429.75
27. Electricity	8	.17	34	4.25
28. Explosives	10	.21	60	6.00
29. All other surface accidents	61	1.28	11,252	184.46
Total surface	434	9.12	77,320	178.16
Cause unknown	1	.02	128	128.00
Grand total	4,756	100.00	1,186,456	249.47

TABLE 5. - Compensable and noncompensable accidents in 150 Colorado coal mines, 1929-33 classified according to parts of the body injured

	Head injuries	Eye injuries	Total or partial loss of eye	Hand and finger injuries	Foot injuries	Hernia	Miscellaneous injuries
1929							
Number of cases	18	97	19	197	155	12	494
Days lost	453	20,417	19,620	16,641	11,902	1,128	313,350
Compensation	$662.13	$14,324.62	$13,829.57	$10,316.91	$12,064.57	$3,051.87	$244,374.97
Days lost per case	25.17	210.48	1,032.63	84.47	76.79	94.00	634.31
1930							
Number of cases	28	93	17	182	139	14	408
Days lost	18,350	17,183	16,578	11,478	17,434	1,186	206,091
Compensation	$5,672.40	$11,154.87	$10,900.16	$6,898.34	$13,449.30	$1,676.29	$178,703.25
Days lost per case	655.36	184.76	975.18	63.07	125.78	84.71	505.12
1931							
Number of cases	45	111	10	203	164	16	451
Days lost	8,496	11,570	13,380	17,224	13,610	677	122,517
Compensation	$3,457.56	$7,704.78	$8,457.63	$10,765.23	$11,744.86	$1,053.00	$118,712.68
Days lost per case	188.80	104.23	1,338.00	84.85	82.99	42.31	271.66
1932							
Number of cases	32	110	12	209	151	20	480
Days lost	12,371	13,977	13,320	15,489	4,662	1,294	170,287
Compensation	$5,171.00	$10,733.13	$10,463.96	$8,182.54	$3,895.19	$1,471.29	$122,466.12
Days lost per case	386.59	127.06	1,110.00	74.11	30.87	64.70	354.76
1933							
Number of cases	34	120	7	212	156	23	467
Days lost	18,526	9,140	10,170	14,302	8,210	3,929	112,726
Compensation	$10,040.51	$4,751.68	$5,191.90	$7,659.86	$6,704.19	$5,050.36	$93,158.11
Days lost per case	544.88	76.17	1,452.86	67.46	52.63	170.83	241.38
Total							
Number of cases	157	531	65	1,003	765	85	2,300
Days lost	58,196	72,287	73,068	75,134	55,868	8,214	924,971
Compensation	$25,003.60	$48,669.08	$48,853.22	$43,322.88	$47,858.11	$12,302.81	$757,415.13
Days lost per case	370.68	136.13	1,124.12	74.91	73.03	96.64	402.16

TABLE 6. - Cost of fatal and permanent partial and total disability accidents in 150 Colorado coal mines, 1929-33

Year	Fatalities	Compensation cost (includes funeral)	Cost per case	Permanent total disabilities	Compensation cost	Cost per case	Permanent partial disabilities	Compensation cost	Cost per case	Total compensation
1929	37	$98,276.23	$2,656.11	3	$63,365.12	$21,121.71	98	$92,453.37	$943.40	$254,094.72
1930	27	68,619.53	2,541.46	2	37,040.08	$18,520.04	82	89,226.32	1,088.13	194,885.93
1931	12	31,416.18	2,618.02	2	22,906.68	11,453.34	83	72,496.64	873.45	126,819.50
1932	21	47,124.39	2,244.02	1	5,252.00	5,252.00	75	78,164.73	1,042.20	130,541.12
1933	13	31,910.13	2,454.63				75	71,275.78	950.34	103,185.91
Total	110	$277,346.46	$2,521.33	8	$128,563.88	$16,070.48	413	$403,616.84	$977.28	$809,527.18

TABLE 7. - Fatalities in Colorado coal mines by cause, 1929-33

Cause	1929	1930	1931	1932	1933	Total
1. Falls of roof (rock, coal, or draw slate)	20	15	5	11	5	56
2. Falls of roof due to car or machine knocking out post	1					1
3. Falls of face or rib	1		1			2
6. Falls of persons	2	1				3
11. Mine cars and locomotives						
A. Struck, run over, or squeezed between	4	2	2	8		16
B. Squeezed between car and rib, timber or roof	1	4	1			6
C. Derailments	1				1	2
F. Animals on haulage		1		1		2
G. Falling from cars (not run over)				1		1
12. Explosions of gas or coal dust			1			1
13. Explosives (not including explosions of gas or dust)	1					1
14. Electricity (not resulting in explosions)	4	1			2	7
15. Machinery (not including 12 and 14)						0
A. Mining machines		1				1
C. All other					1	1
17. Mine fires (burns, suffocations etc.)					2	2
19. Vertical or inclined shaft			1			1
21. Railway cars and locomotives	1	1				2
24. Falls of persons		1			1	2
25. Falling objects	1					1
26. Machinery			1			1
29. All other surface accidents					1	1
Total	37	27	12	21	13	110

TALBE 8. - Average compensation for compensable temporary disability in Colorado coal mines, 1929-33

Year	Number of cases	Days lost	Days lost adjusted for 10-day waiting period	Compensation paid	Compensation[1] per day
1929	483	21,779	16,949	$27,648.48	1.63
1930	396	17,089	13,129	20,992.23	1.60
1931	461	21,686	17,076	25,565.61	1.50
1932	441	20,160	15,750	19,906.86	1.26
1933	443	20,610	16,180	19,128.44	1.18
Total	2,224			$113,241.62	
Average per case				50.92	

1 Average compensation per day of temporary disability, excluding the 10-day waiting period for which no compensation is paid.

TABLE 9. - Scale of time losses for weighting industrial accident disabilities to show severity of accidents, according to the Association of Industrial Accident Boards and Commissions

Nature of injury	Degree of disability in percent of permanent total disability	Days lost
Death	100	6,000
Permanent total disability	100	6,000
Arm above elbow, dismemberment	75	4,500
Arm at or below elbow, dismemberment	60	3,600
Hand, dismemberment	50	3,000
Thumb, any permanent disability of	10	600
1 finger, any permanent disability of	5	300
2 fingers, any permanent disability of	12½	750
3 fingers, any permanent disability of	20	1,200
4 fingers, any permanent disability of	30	1,800
Thumb and 1 finger, any permanent disability of	20	1,200
Thumb and 2 fingers, any permanent disability of	25	1,500
Thumb and 3 fingers, any permanent disability of	33-1/3	2,000
Thumb and 4 fingers, any permanent disability of	40	2,400
Leg above knee, dismemberment	75	4,500
Leg at or below knee, dismemberment	50	3,000
Foot dismemberment	40	2,400
Great toe or any 2 or more toes, any permanent disability of	5	300
1 toe, other than great toe, any permanent disability of	0	---
1 eye, loss of sight	30	1,800
Both eyes, loss of sight	100	6,000
1 ear, loss of hearing	10	600
Both ears, loss of hearing	50	3,000

United States Bureau of Mines Publications on Accident Costs

1. CRAWFORD, F. S., The Cost of Accidents to Industry. Inf. Circ. 6333, 1930, 10 pp.

2. ASH, S. H., Accident Experience and Cost of Accidents in Washington Coal Mines. Inf. Circ. 6529, 1931, 18 pp.

3. MURRAY, A. L., and HARRINGTON, D., Accident Experience of the Coal Mines of Utah for the Period 1918 to 1929. Inf. Circ. 6530, 1931, 26 pp.

4. FENE, W. J., Accident Experience and Cost in Pennsylvania Anthracite and Bituminous mines, 1926-30. Inf. Circ. 6618, 1932, 29 pp.

5. CASH, F. E., Accident Experience and Cost in Tennessee Coal Mines. Inf. Circ. 6664, 1932, 8 pp.

6. HERBERT, C. A., Ten years of Fatal Accidents and Two Years of Accident Costs in Indiana Coal Mining. Inf. Circ. 6672, 1932, 12 pp.

7. ASH, S. H., Accident Experience and Cost of Accidents at Washington Metal Mines and Quarries. Tech. Paper 514, 1932, 35 pp.

8. PARKER, D. J., Accident Experience and Costs in Wyoming Coal Mines. Inf. Circ. 6791, 1934, 13 pp.

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10. DAVIES, J. F., and HUMPHREY, H. B., Accident Experience and Cost in Virginia Coal Mines. Inf. Circ. 6763, 1933, 15 pp.

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ACCIDENT EXPERIENCE AND COST IN CALIFORNIA METAL MINES

BY

S. H. ASH

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

ACCIDENT EXPERIENCE AND COST IN CALIFORNIA METAL MINES[1]

By S. H. Ash[2]

CONTENTS

INTRODUCTION

For several years considerable activity[3] has been devoted to economic and social factors affecting California metal-mining operations as they relate to accidents and their cost. The reason for this intense interest is that accidents cost this industry annually over $400,000 in industrial insurance premiums for mines insured by insurance carriers only; during 1924-32, inclusive, they amounted to $3,616,945. There are several reasons for this high cost, the most important of which are the consistently high frequency and severity rates of injuries occurring in the industry. Coordination of effort, appreciation of a mutual problem, and widespread education are essential if accidents and their cost are to be reduced.

1 The Bureau of Mines will welcome reprinting of this Information Circular provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6861."

2 District engineer, U.S. Bureau of Mines Safety Station, Berkeley, Calif.

3 Ash, S. H., Explosives Accidents in California Metal Mines: Inf. Circ. 6725, Bureau of Mines, 1933, pp. 1-2.
Glaeser, Oscar A., Accident Prevention in Metal Mines, with Special Reference to Gold Mining: Min. Cong. Jour., December 1934, pp. 18-21. Presented at Meeting of West. Div., Am. Min. Cong., San Francisco, Calif., Sept. 26, 1934.
California Journal of Development, State Chamber of Commerce, September 1934.

This report points out certain factors in accident experience and mining practice that affect accident occurrence and cost to the metal-mining industry of California.

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STATUS OF THE INDUSTRY

Gold mines predominate in California; many of them are small, relatively short-lived, and scattered over the State, usually in isolated sections. A mine employing 10 men is now considered a large mine in California, the leading gold-producing State in the United States. In 1933[4] (table 2) California ranked first in the number of men employed (7,892) and man-hours of labor performed (13,226,793) among the metal-mining States of the United States, followed by Michigan and Minnesota. California leads in the number of operating mines, with 973 for 1933. Accident-prevention difficulties compared with those of some States are apparent when the number of mines is considered - Michigan and Minnesota combined reported 134 mines for the same year. Except for 1930 the number in California reporting to the Industrial Accident Commission has steadily increased from 318 in 1924 to 973 in 1933 (more than 200 percent), and the average number employed per mine has decreased from 19.6 in 1924 to 8.1 in 1933. These facts explain in part the increasing cost of insurance, inspection, and supervision and indicate the necessity for greater emphasis on safety education and increased supervision.

Accident costs have increased with the increase in number of operations and the decrease in number of employees in the average mine. The classification rate per $100 of pay roll for mining was $5.81 in 1924 and $11.85 in 1933. Revision of classification reduced this rate to $11 in 1934 and 1935. This does not mean that the total amount paid by the industry will be reduced, for unless

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4 Adams, W. W., Metal-Mine Accidents in the United States During the Calendar Year 1932: Bull. 377, Bureau of Mines, 1934, p. 5.
Accident-Prevention Record of the Metal-Mining Industry in the United States in 1933, Including Nonmetallic Mineral Mining Other than Coal: Health and Safety Statistics 185, Demographical Division, Bureau of Mines, Dec. 6, 1934.

5 Ash, S. H., Explosives Accidents in California Metal Mines: Inf. Circ. 6725, Bureau of Mines, 1933, pp. 1-2.

accident costs were reduced in 1934 the excess cost was merely distributed in the new classifications and must be paid in the end. The encouraging fact is that the rate for 1935 remains unchanged, which shows that accident-prevention work apparently is producing results. A review of fatalities for 1934, however, shows that unless efforts are intensified the desired relief is not in sight insofar as human suffering is concerned.

ACCIDENT EXPERIENCE

For many years the United States Bureau of Mines has been compiling and publishing annually statistics on accidents in the mineral industries. Prevention of accidents requires a knowledge of the exposure to which the workmen are subjected; number, frequency, and severity of injuries and time lost therefrom; nature of injuries; and causes of accidents. The nature of injuries often indicates that certain safety-equipment and first-aid practices if applied properly will prevent many types of costly injuries. The same is true to a certain extent of the causes of injuries usually given, which are merely the names of certain physical agencies by which the injuries are inflicted. Complete accident reports should therefore be available so the interested, safety-conscious management will see that the real causes of accidents involve mining practice, equipment, and human factors.

The tables given compare the accident experience in California metal mines over a period of years. The accident rates are based on man-hours of exposure, and accident-frequency (number of accidents per million man-hours of exposure) and accident-severity (time lost per thousand man-hours) rates are used. These tables include all lost-time injuries, although for compensation-cost purposes California has an absolute 7-day waiting period which is included in all lost-time data.

Table 1 shows the number of injuries and time lost for all lost-time injuries by class of injury at all California metal mines for 1929-33, inclusive.

Table 2 shows the number of mines, man-hours of exposure, and accident-frequency, fatality, and injury rates in California metal mines for 1924-33, inclusive.

In tables giving data for all mines Bureau of Mines classifications [6] are used; these include self-insurers and other groups not included in risks in classification 1164, Mining, N.O.C., held by the California insurance carriers.

6 Adams, W. W., Metal-Mine Accidents in the United States During the Calendar Year 1928: Bull. 320, Bureau of Mines, pp. 4, 63.

TABLE 1.- Number of injuries and days lost for all lost-time injuries by class of injury at all California metal mines, 1929-33, inclusive[1]

Class of injury	1929		1930		1931		1932	
	No. of injuries	Days lost	No. of injuries	Days lost	No. of injuries	Days lost	No. of injuries	Days lost
Killed	31	186,000	29	174,000	15	90,000	25	150,000
Permanent total disability	------	-------	------	-------	------	-------	------	-------
Permanent partial disability	22	17,600	25	20,000	23	18,400	18	14,400
Temporary disability	1,426	19,476	1,386	18,018	1,137	14,781	911	11,843
All nonfatal	1,448	37,076	1,411	38,018	1,160	33,181	929	26,243
Grand total	1,479	223,076	1,440	212,018	1,175	123,181	954	176,243

Class of injury	1933		1924-28		1929-33	
	No. of injuries	Days lost	No. of injuries	Days lost	No. of injuries	Days lost
Killed	9	54,000	129	774,000	109	654,000
Permanent total disability	1	6,000	2	12,000	1	6,000
Permanent partial disability	21	16,800	174	139,200	109	87,200
Temporary disability	1,558	20,254	8,156	111,752	6,418	84,372
All nonfatal	1,580	43,054	8,332	262,952	6,528	177,572
Grand total	1,589	97,054	8,461	1,036,952	6,637	831,572

1 Compiled from Bureau of Mines reports on metal-mine accidents in the United States.

TABLE 2.- Accident and frequency rates and man-hours of exposure in California metal mines, 1924-33, inclusive 1/

Year	Number of mines	Men employed	Man-hours worked	Killed	Injured	Rate per million man-hours Killed	Rate per million man-hours Injured
1924	318	6,238	15,440,784	32	1,840	2.07	119.17
1925	344	7,078	17,129,472	30	1,870	1.75	109.17
1926	415	6,849	16,522,760	38	1,793	2.30	108.52
1927	453	6,584	14,465,112	19	1,548	1.31	107.02
1928	496	6,433	13,867,784	10	1,281	.72	92.37
1929	538	7,455	15,297,376	31	1,448	2.03	94.65
1930	476	5,941	11,908,896	29	1,411	2.43	118.48
1931	640	5,553	10,755,493	15	1,160	1.39	107.85
1932	817	5,646	10,051,955	25	929	2.49	92.42
1933	973	7,892	13,226,793	9	1,580	.68	119.45
1924-28	2/405	2/6,636	77,425,912	129	8,332	1.67	107.61
1929-33	2/689	2/6,498	61,240,513	109	6,528	1.78	106.60

1 Compiled from Bureau of Mines reports on metal-mine accidents in the United States.

2 Average.

According to data in table 2 the accident rates for the two 5-year periods 1924-28 and 1929-33 virtually have stood still, the fatality severity rate showing an increase during the last 5-year period due to the mine-fire catastrophe in 1930 and the "high explosives" and "falling-in-shaft" accidents during this period.

A careful review[7] of all mine-accident statistics for California shows that the fatal-accident rates are a more reliable indication of the trend of safety performance over a long period than relatively recent rates that include non-fatal temporary and partial-disability accidents that enter directly into compensation insurance awards. Table 2 shows emphatically the necessity for changes in mining practice relating to safety, as the accident occurrence as well as accident rates are unduly high and show little if any tendency to improvement.

Table 3 gives the frequency and severity rates for all accidents at all mines during 1924-33. The Bureau of Mines schedule of time lost for injuries[8] has been used in accordance with the scale adopted by the Association of Industrial Accidents Boards and Commissions, which is -

7 Ash, S. H., Mine Safety Practices: California Jour. of Development, State Chamber of Com., September 1934, pp. 16, 46.
Ash, S. H., Explosives Accidents in California Metal Mines: Inf. Circ. 6725, Bureau of Mines, 1933, pp. 1-2.

8 Adams, W. W., Metal-Mine Accidents in the United States, 1930: Bull. 362, Bureau of Mines, 1932, p. 62.

	Weighted at, days
Death	6,000
Permanent total disability.	6,000
Permanent partial disability.	800
Temporary disabilities (1 day or more) . . .	13

Table 4 shows the frequency and severity rates for all lost-time injuries by class of injury at all California mines for 1929-33, inclusive, and the average for the 5 years 1924-28.

A review of frequency data alone of all lost-time accidents often leads to misconception of safety work and accident-cost rates. Unquestionably the accident-frequency rate which is so readily available furnishes the "prodding" tool for the safety-conscious mine manager, but the severity of injuries is the factor that indicates the hazard of the industry, the extent of human suffering, and the cost of accidents.

TABLE 3.- Man-hours worked and accident frequency and severity rates at all metal mines in California, 1924-33 [1]

Year	Man-hours worked	Accident-frequency rate	Accident-severity rate
1924	15,440,784	121.24	16.844
1925	17,129,472	110.92	13.733
1926	16,522,760	110.82	17.182
1927	14,465,112	108.33	10.935
1928	13,867,784	93.09	7.130
1929	15,297,376	96.68	14.582
1930	11,908,896	120.91	17.814
1931	10,755,493	109.24	11.454
1932	10,051,955	94.91	17.535
1933	13,226,793	120.13	7.326
1924-28	77,425,912	109.278	13.393
1929-33	61,240,513	108.386	13.569

1 All lost-time accidents are taken from Bureau of Mines bulletins on metal-mine accidents in the United States.

Analysis[9] of accident rates indicates that the frequency of fatal accidents over a long period is of more value in projecting the severity of all accidents than the frequency figure for all nonfatal accidents. The significance of this statement is indicated in table 4 by the frequency (1.78) of fatalities and the severity (13.57) for all injuries for the period 1929-33, which are greater than the frequency (1.67) and the severity (13.39), respectively, for the 5-years 1924-28; as regards human suffering and economic loss the later period was worse. Economically this is reflected in increased compensation insurance rates, as indicated by the cost rates at the close of each period.

9 Ash, S. H., Accident Prevention and Statistics in Tunneling Operations: Presented at the meeting of the General Safety Committee of the Metropolitan Water District of Southern California, Oct. 8, 1934.

TABLE 4.- Frequency and severity rates for all lost-time injuries by class of injury at all California mines, 1929-33, inclusive

Class of injury	1929		1930		1931		1932	
	Frequency	Severity	Frequency	Severity	Frequency	Severity	Frequency	Severity
Killed	2.03	12.16	2.43	14.62	1.39	8.37	2.49	14.92
Permanent total disability	------	--------	------	--------	------	--------	------	--------
Permanent partial disability	1.44	1.15	2.10	1.68	2.14	1.71	1.79	1.43
Temporary disability[1]	93.21	1.27	116.38	1.51	105.70	1.37	90.63	1.18
All nonfatal	94.65	2.42	118.48	3.19	107.85	3.08	92.42	2.61
Grand total	96.68	14.58	120.91	17.81	109.24	11.45	94.91	17.53

Class of injury	1933		1924-28		1929-33	
	Frequency	Severity	Frequency	Severity	Frequency	Severity
Killed	0.68	4.07	1.67	10.00	1.78	10.67
Permanent total disability	.08	.45	.03	.15	.01	.10
Permanent partial disability	1.59	1.27	2.25	1.80	1.78	1.42
Temporary disability 1/	117.78	1.53	105.33	1.44	104.81	1.38
All nonfatal	119.45	3.25	107.61	3.39	106.60	2.90
Grand total	120.13	7.32	109.28	13.39	108.38	13.57

1 Disability for more than remainder of day of accident.

On the other hand, the total frequency figure dropped from 109.28 to 108.38. The remedy for reducing severity lies in reducing the frequency of serious accidents resulting from falls, blasting, falling down shafts, suffocation, and fires, as well as other infrequent causes.

Comparison of accident rates with accident costs shows that a high total severity rate for any one year is not always reflected in higher costs, as fatal accidents, although of maximum severity, cost less than some types of serious nonfatal injuries because of the comparatively low financial remuneration for a fatality which is based on dependency and also is limited (table 9, p. 24). The relative value of the compensation item of accident costs projected from the severity rate in California is in the following order: (1) Compensable nonfatal, (2) all lost-time nonfatal, (3) all injuries, and (4) fatalities.

Table 5 gives fatalities by causes at all California metal mines for 1928-32.

TABLE 5.- Fatalities by causes at all California metal mines for the calendar years 1928-32, inclusive

Cause	1928	1929	1930	1931	1932	1928-32 Total	1928-32 Percentage of total
Underground:							
Falls of rock or ore from roof or wall....	3	13	6	7	3	32	29.09
Timber or hand tools....................	----	----	2	----	----	2	1.82
Explosives..............................	----	6	5	2	9	22	20.00
Haulage.................................	----	----	3	1	----	4	3.64
Falling down chute, winze, raise, or stope	1	1	1	3	1	7	6.37
Run of ore from chute or pocket..........	----	1	----	----	----	1	.91
Electricity.............................	----	----	----	----	1	1	.91
Machinery other than locomotives or drills	1	----	1	----	----	2	1.82
Mine fires..............................	----	----	5	----	----	5	4.54
Suffocation from natural gases...........	1	2	----	----	3	6	5.45
Inrush of water.........................	----	----	1	----	----	1	.91
Subtotal...........................	6	23	24	13	17	83	75.46
Shaft:							
Falling down shaft......................	----	4	1	----	1	6	5.45
Objects falling down shaft..............	----	----	1	----	1	2	1.82
Breaking of cables......................	----	----	----	2	----	2	1.82
Skips, cages, or buckets................	1	3	2	----	2	8	7.27
Subtotal...........................	1	7	4	2	4	18	16.36
Surface:							
Electricity.............................	----	----	----	----	2	2	1.82
Machinery...............................	1	----	----	----	1	2	1.82
Handling materials......................	----	----	1	----	1	2	1.82
Other causes............................	2	1	----	----	----	3	2.72
Subtotal...........................	3	1	1	----	4	9	8.18
Grand total........................	10	31	29	15	25	110	100.00

Although statistical data for previous years are necessary for projecting accident experience, yet the same accidents are happening now in much the same way and from the same causes because certain mining practices conducive to accidents remain unchanged for the industry as a whole. As accident-prevention work progresses in certain mines (and it has considerably in California) with beneficial results, in other less safety-conscious mines conditions remain the same, and the sum total of compensable accidents is the same or only slightly unchanged. A review of the fatalities at all metal mines in California during 1934, coming to the attention of the United States Bureau of Mines, is therefore presented. These data are tentative and subject to revision.

Fatality 1

A gold-dredge shore boss, whose age was not given, received injuries about the head and died $2\frac{1}{2}$ months later. He was struck by a "come along." Foremen should be more familiar with cable hazards[10] than other employees, but far too frequently they are inclined to "take chances" for which they would condemn their employees.

Fatality 2

A timberman 33 years old received an eye injury which later became infected and caused his death about $2\frac{1}{2}$ months later. This accident was caused by a small fragment of rock flying from a rock which a fellow workman was breaking with a hammer. Such injuries can be prevented by wearing proper goggles. The danger from infection of injuries about the head is emphasized by this fatal accident.

Fatality 3

A 22-year-old shop laborer of a gold-dredge crew was electrocuted when he contacted a high-tension wire while assisting the surveyor. Familiarity with the hazards of the surroundings and care in avoiding them can prevent this type of accident, which is not uncommon when a workman follows other than his regular occupation.

Fatality 4

A miner 40 years old was killed at a gold mine when struck on the head by a piece of rock falling down the manway of a raise. The man was climbing the ladder in the manway of the raise at the beginning of the shift to investigate the amount of rock to be drawn from the chute while his partner had gone after a car. A rock weighing about 8 pounds, probably lying in the manway, became dislodged in some manner and struck him on the head.

Head protection with any of the hats now available might have saved this life. This is required at most of the large mines in California and should be required of all workmen at all mines, large or small.

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10 Ash, S. H., Safety Practices in California Gold Dredging: Bull. 352, Bureau of Mines, 1932, 31 pp.

Fatality 5

A miner 48 years old was asphyxiated from powder gas in a gold mine. Five men, working this mine under a "partnership", were driving a drift which had progressed about 250 feet from a shaft at the 250-foot level; the shaft dips about 70°.

A round of 14 holes using 70 sticks of $1\frac{1}{4}$- by 8-inch gelatin dynamite, with fuse and caps, had been blasted in the drift at 5 p.m., when the men went off the shift; about 7:45 p.m. the deceased and another man went down the shaft to inspect the result. About 10 p.m. a brother of one of the men went to see why they had not returned; a short distance in he found one of the men apparently dead, and farther in he found his brother unconscious. He dragged both men to the landing and after obtaining assistance took the injured men to the surface and sent for a doctor. When the doctor arrived one man had revived, but efforts to revive the other failed.

This is another example of an accident resulting from lack of ventilation and safe practice at a "partnership" mine. Had the rescuers been familiar with first-aid resuscitation methods one man could have been revived quickly (he revived in fresh air), and perhaps the other man could have been resuscitated. However, the man who revived without treatment was the only one who knew how to administer first aid. This accident illustrates the value of 100-percent first-aid training in which all persons at a plant take the full Bureau of Mines course of instruction in first aid.

These men are said to have thought there was no danger from the fumes of the explosive they were using, basing their belief upon a notation enclosed in the boxes used for the explosive.

The compressed air was insufficient to clear the place in time. This accident shows plainly the lack of intelligent supervision and need for better ventilation[11] in some of the metal mines of this country.

Fatality 6

A mule driver 39 years old was killed at a nonmetallic mineral mine when crushed between a car and the ribs of the drift. He lost control of the mule hauling the car and was squeezed between the car and the side wall. The deceased left a widow and seven children.

This type of accident is common when men ride on the front car bumper or on the tail chain instead of riding the back bumper or if possible inside an empty car and suggests at least more clearance for such haulage.

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11 Harrington, D., and Denny, E. H., Gases that Occur in Metal Mining: Bull. 347, Bureau of Mines, 1931, 21 pp.

Fatalities 7 and 8

A miner 38 and a mucker 22 years old were killed by a slide of rock at a gold mine. At the time of the accident the two men were "cleaning-up" and pulling out old timbers from a shrinkage stope and were buried by a slide. Evidently the timbering was inadequate, and when timbers were removed the rock was released. Shrinkage stopes present many hazards due to lack of timbering, and at the time of cleaning it often is impossible to test the walls. Square-sets and even cribbing often are advisable in questionable ground. Careful inspection and precautionary measures to control the ground will prevent such accidents.

Fatality 9

A timberman 33 years old was killed at a gold mine by a fall of hanging-wall rock. He was removing timbers to build a chute; when a stull was removed a slab of rock slid from the hanging wall, breaking another timber and crushing the timberman. Evidently removal of the stull weakened the ground, and the deceased erred in not properly securing the hanging wall or used poor judgment in testing the overhanging material.

Fatality 10

A miner 65 years old was killed at a small gold mine by flying rock from a blast. He and his partner were the only workmen at this mine. They had drilled and loaded a round of 7 holes in a crosscut they were driving and were lighting the fuses with their carbide lights. The deceased had just renewed the carbide charge in his lamp, and the gas pressure was so strong the lamp would not stay lighted. He reached over to take a light from his partner's lamp but in doing so extinguished the partner's light also, leaving them in darkness. They started to walk in the dark, following the rails of the track to a point about 80 feet away where the crosscut intersected a drift. A mine car was on the track between the face and the drift around which they had to pass. The deceased's partner reached the drift, where he waited, and presently the shots went off. He succeeded in lighting his lamp and not seeing his partner started for the face and found him lying stretched out toward, and about 40 feet from, the face. He was dead, with a rock about the size of an egg embedded in his skull. It is thought that he lost his sense of direction in going around the car and was walking back toward the face in the dark when the blast went off. Neither of the men wore head protection which, while not intended for cases of this kind, nevertheless has saved lives under similar circumstances.

This accident is typical of an alarming number from blasting that continue to occur in California mines. It could have been prevented if the men had been provided with electric cap lamps, or if another lighted lamp had been in the drift, or if some method of electric blasting had been used.

This type of accident, caused in this instance by experienced men, is one of the main reasons for high accident rates and costs in California.

Fatality 11

A miner 50 years old was killed at a gold mine when he walked into a blast in which fuse and caps were used. At the time of the accident some holes were being blasted on a road job leading to the mine. The deceased, an experienced miner, is said to have paid no attention to the blasting signals or personal warning but walked directly into the blast.

This accident also is more or less typical; the deceased ignored the warnings, misunderstood them, or did not hear them. Many accidents of this type can be prevented by electric blasting whereby the moment of firing is in control at essentially all times.

Fatality 12

A miner 36 years old was killed at an underground silica mine by an air blast from a cave. After shooting 2 rounds of holes intended to cave 2 weak pillars and the capping over some old stopes the deceased started to enter one of the stopes almost immediately after the shots had been fired; he was traveling from the surface through a raise connecting with the stope when a cave occurred (capping fell). The force of the resulting air blast hurled him several hundred feet, causing fatal injuries.

This type of accident is not uncommon and reveals an error in judgment, as air blasts while by no means numerous can be expected under such circumstances. Many workmen are killed, directly or indirectly, by entering working places too soon after blasting, particularly when blasting is done at any time during the working shift. Such accidents can be avoided by blasting at the close of the shift, if at all feasible; by keeping the face clear of workmen for some time; by closer supervision; and by the exercise of better judgment by the workman himself. No one unfamiliar with such work should do blasting.

Fatality 13

A mucker 21 years old was killed at a gold mine by a fall of rock in a stope. At the time of the accident ore was being drawn from the stope through the chute. The man was crossing the stope when a slab of rock from the hanging wall struck him. The roof rock, evidently loose, probably was weakened by the removal of some of the loose ore. It may have been working loose, and the deceased did not hear it owing to the noise. Similar accidents can be avoided by careful testing and closer inspection of the roof and by taking down, barring or otherwise, or timbering dangerous rock.

Fatality 14

A miner 48 years old was hit by a fall of rock at a gold mine and died from the injuries 1 week later. This accident occurred at the working face while the deceased was drilling his round. Two holes had been drilled, and

the miner was drilling the third when a slab of rock fell from the back and struck him on the head. Death resulted from concussion of the brain, as an X-ray showed no fracture. It is stated that the deceased was wearing a "hard-boiled" hat at the time and also had barred down. He left a widow and 10 children.

An accident of this type is not uncommon during drilling. Rock that appears safe and resists barring down becomes loose through jarring of the drill; this emphasizes the necessity of testing the roof frequently during drilling operations. Moreover, too much reliance should not be placed on "head protection", which has its limitations although it has prevented and will prevent many serious and minor injuries.

Testing the overhead material in mine workings is one of the most important factors in preventing injuries from falls of overlying material and should be done systematically and often. Experienced miners can thus determine the security of the overlying material and know when to take down dangerous material or to set timber to secure it against falling.

Two methods generally are used to test the back; each involves striking the overlying material with a wooden or metal tool or rod. Striking the back produces sound and vibration. A "drummy" or hollow sound usually indicates that the material is loose, but absence of sound does not always indicate that it is safe. Vibration usually can be felt by the bare fingers or hand in contact with the material being tested. The amount of vibration produced when the material is struck is a better indication of the firmness or looseness of the material; hence, the vibration method should be used where at all feasible. By this method the back or roof material is struck with a rod held in one hand, while the bare fingers of the other are held against the material. The person doing the testing thus has the benefit of both sound and vibration. Vibrations that are easily felt indicate loose material, which should be taken down or supported by timber. Some large pieces of loose roof material may vibrate only slightly if at all and make no drummy sound; this condition is unusually hazardous and shows the necessity of using systematic timbering if adequate protection is to be afforded at all times and under essentially all conditions.

Fatality 15

A millman 65 years old was killed at a gold mine while oiling the machinery in the ore mill. His clothing caught in the drive shaft of the machinery which broke his neck and back and caused other injuries.

The remedy for preventing such accidents is the proper guarding of machinery; if it cannot be guarded it should be fenced off; and if for any reason it is necessary to remove guards or to oil unguarded the machinery should be stopped and not started again until the oiler in person reports this fact to the operator of the driving engine or mechanism. Men working around machinery with moving parts should be so dressed that few if any parts of the clothing are loose enough to be caught in the machinery.

Fatality 16

A skip tender 43 years old was killed at a gold mine while riding on a water skip in the course of his duties. The skip jumped the track and caught his leg between the bail and the skip, fracturing his leg at the femur and severing an artery; he bled to death.

This life undoubtedly could have been saved by proper first-aid treatment -- an instance indicating the necessity for 100-percent first-aid training at all mines.

The accident might have been prevented by some arrangement to dispense with riding skips. Water skips usually do not run under the same conditions as ore skips. Frequently they are run into poorly tracked sumps at high speed; water spilling from the skip often washes material onto the track and causes derailments; and maintenance is often neglected.

Fatality 17

A miner 36 years old was killed and a fellow workman injured by a fall of back at a gold mine. The men were timbering a section of the back near the face in a long raise with a treacherous hanging wall when a portion of it broke.

Such accidents can be prevented only by systematic timbering and careful testing of the overlying material. Accidents during timbering operations are avoidable; the fact that the back requires support emphasizes the necessity for more care in working under dangerous overhead material.

Fatality 18

A filter tender 24 years old was fatally burned by hot solution in a nonmetallic mineral mine. He was closing the valves of an Oliver filter in the surface evaporation building when a hose connection burst and the hot liquid issuing from the break burned him.

Accidents due to failure of pipe lines, hose or their accessories, often show evidence of faulty equipment on careful inspection. They can be prevented by care in operating connections or valves so as not to cause strains, which may result in ruptures. Relief valves should be properly placed to take care of contingencies, and instructions should be issued and enforced regarding proper operation.

Fatality 19

A mine foreman 61 years old was killed at a gold mine while riding to the surface on an ore skip; he was caught between the skip and the chute spout at a station, and his head and chest were crushed.

This common shaft accident can be prevented by riding properly in skips. Mine foremen should be more familiar with dangers in a shaft than other workmen. It is unfortunate that officials far too frequently ride haulage equipment under conditions so hazardous as to be prohibited for other workmen; in addition to the danger to officials, this foolhardiness sets a bad example and encourages other workmen to disobey regulations.

Fatalities 20 and 21

Two miners 43 and 45 years old, respectively, were killed by a fall of roof at a gold mine. These men were working in an open cut-and-fill stope; a change of ground occurred, and there was no timber in the stope. A large portion of the overlying material caved without warning and crushed the two men.

This accident shows the danger of placing too much dependence upon so-called "good back." Safe conditions are taken for granted, with disastrous results sooner or later. Proper testing of the roof or back and adequate timely timbering would have prevented this accident.

Fatality 22

A miner 37 years old died from injuries received from a falling timber in the course of his duties at a gold mine. His leg was fractured, and he died from pulmonary embolism.

Accidents from this cause often result if the timber is handled improperly, knocked out by some other agency, or released by some other workman, or falls off trips while being hauled. Determining the remedy is relatively simple, but seeing that it is applied at all times is a wholly different matter.

Fatality 23

A miner 32 years old died the day after injuries received from a fall of rock while working underground at a gold mine.

No other details of this accident were available.

Fatality 24

A partnership miner 44 years old was killed by a slide of rock at a surface placer mine. He had completed his work and sat down to rest in a cut in the gravel bank in which he had been working. Rain had loosened the ground, and a rock broke loose and crushed the deceased. A newspaper item referred to this fatality as a "freakish" accident. There is nothing "freakish" about a bank sliding in; it can be anticipated in any freshly dug, unsupported trench. Such accidents often occur in small prospects in the absence of responsible supervision and adequate safety precautions. It is essentially as important to rest in a safe place as to work in one.

Fatality 25

A miner 40 years old was killed at a gold mine by a fall of rock. In this instance the deceased had sounded the roof and picked it down preparatory to drilling. He drilled the round and was taking the drill bar back in the stope when a slab fell and crushed his body.

This accident is similar to fatality 14; the jarring of the drill evidently loosened the back. Failing to test the roof or back frequently, the man was unaware of its condition.

Fatality 26

A mine foreman 48 years old was asphyxiated from powder gas at a gold mine. Three rounds had been blasted in a raise, and the deceased and another man went into the raise to inspect the results of the blasts. The foreman was totally overcome with gas, and the other man was incapable of giving assistance but managed to get out. Persons with a knowledge of first aid were not available to give assistance, again indicating that all persons employed in mining should be required to have an adequate knowledge of first-aid work.

This accident, similar to fatality 5, emphasizes the need of ventilation, more intelligent supervision, and knowledge of first aid at small metal mines. The excessive use of explosives, aside from causing fatalities, contributes much to the ill health of many miners, particularly when they go into the gassy dust-laden air too soon after blasting.

Fatality 27

A mucker 28 years old was killed by a fall of roof in a gold mine. In this instance the man was mucking out an old tunnel preparatory to retimbering; the walls caved, crushing his chest and skull.

This accident indicates lack of proper supervision, inspection, and support of the ground to be retimbered; in all probability removing the muck disturbed the ground and old timbers and permitted the ground to cave.

Fatality 28

A miner at a "partnership" gold mine was killed by a fall of rock. Two men were opening an old mine; one of them decided to go to an adjacent old mine and get some timber. Removing the timber caused a cave, with fatal results.

Such conditions are likely to prevail at small mines and prospects being opened for sampling or possible operation; they contribute to offset increased safety at operating mines and also affect the accident record of the whole industry.

Fatalities 29 and 30

A miner was asphyxiated instantly, another miner died later, and one was made seriously ill at a gold mine by fumes from a dynamite explosion. The explosion occurred from an unknown cause when the deceased miner entered an underground magazine against orders. As a result of the blast another man was asphyxiated in leaving his nearby working place and, although rescued, died later from the effect of the fumes. His rescued partner recovered.

This accident shows how one man's mistake or carelessness may endanger the lives of others. Another workman who was responsible for saving one of the men risked his life in a desperate attempt to save both.

Compliance with well-considered rules and regulations will prevent such accidents.

Fatality 31

A miner 21 years old was killed when he fell down a shaft at a gold mine. The man was descending the ladder in a shaft which dipped 60°; his back was turned toward the ladder when he slipped and fell about 175 feet, fracturing his skull.

Obviously this man was descending the ladder in the wrong manner. Such accidents can be prevented by traveling ladders properly, by having landings at safe intervals where men use ladders, and by providing handrails. Rubber packs or boots, which are not safe for use on ladders, have contributed to the death or injury of many from slipping.

Table 6 shows the number and percentage of fatal injuries by causes for 1934.

Table 7 shows the number of fatalities during 1934 by occupation.

TABLE 6.- Fatal injuries during 1934 by causes

Cause	Fatalities	
	Number	Percentage
Falls of ground:		
Placer gravel bank caved	1	3.23
Fall of rock	1	3.23
Fall of rock while timbering	1	3.23
Fall of rock in stope while pulling ore	1	3.23
Air blast from cave following blasting	1	3.23
Fall of roof on removing muck or timber	3	9.68
Fall of roof in stope or raise	2	6.44
Slides of rock in shrinkage stope	2	6.44
Rock coming down manway	1	3.23
Fall of rock while drilling	2	6.44
Subtotal	15	48.38
Explosives:		
Overcome by powder smoke following blasting	2	6.44
Explosion of dynamite in underground magazine	2	6.44
Struck by flying rock from blast	2	6.44
Subtotal	6	19.32
Haulage:		
Caught between skip and shaft timbers	1	3.23
Caught between bail and skip	1	3.23
Crushed between car and rib	1	3.23
Subtotal	3	9.68
Other causes:		
Slipped and fell down shaft	1	3.23
Struck by "come-along" of cable (surface)	1	3.23
Struck in eye by flying piece of rock from hammer	1	3.23
Electrocuted by high-tension wire (surface)	1	3.23
Struck by falling timber	1	3.23
Clothing caught in mill machinery drive shaft (surface)	1	3.23
Burned by hot liquid at surface cleaning plant	1	3.23
Subtotal	7	22.58
Underground	24	77.42
Surface	7	22.58
Grand total	31	100.00

TABLE 7.- Fatal injuries during 1934 by occupation

Occupation	Fatalities	
	Number	Percentage
Foreman[1] or shift bosses	3	9.68
Miners (under safety act)	13	41.93
Miners (partnership mines)	4	12.90
Muckers	4	12.90
Timbermen	2	6.44
Driver	1	3.23
Skip tender (underground)	1	3.23
Shop helper	1	3.23
Millman	1	3.23
Filter-plant operator (surface)	1	3.23
Total	[2]31	100.00

1 1 killed on surface.
2 7 killed on surface.

SUMMARY OF FATALITIES IN CALIFORNIA METAL MINES, 1934

The 1934 fatality record of 31 deaths, compared with 9 for 1933, was significant in that the exposure in 1934 probably did not exceed if it equaled that of 1933, owing to the sustained suspension of operations at the large mines of the Mother Lode district.

Of the 31 men killed 14 were single men, 13 were married, and 4 were widowers with dependents; 35 orphans and 2 parents of single men were involved in the dependency. The peak months for fatalities were April and July with 5 each. June had no fatalities; March, September, and November had 1 each; January and February 2 each; August, October, and December 3 each; and May 4.

One mine had 3 accidents with 3 fatalities, and 2 mines had 1 accident each with 2 fatalities each. These fatalities occurred at large mines.

The increase of fatalities from rock falls from an average of 33 percent in 1924 to 48.38 percent in 1934 emphasizes the need of closer supervision and inspection and enforcement of timbering safety rules.

Explosives rank second as a cause of accidents with 20 percent of the fatalities -- an unenviable position probably not duplicated in any other mining State.

The 3 (9.68 percent) fatalities among foremen show the high accident frequency and severity of this occupation and reflect unsafe conditions and practices, chance-taking, and especially the need of more thorough safety education for these officials. The 17 miners killed (53.83 percent of the fatalities) emphasize the need of accident-prevention methods at the working face.

CAUSES OF ACCIDENTS

The fatal accidents that occurred during 1934 are fairly typical of mine accidents in California. A comparison of statistics in tables 5, 6, and 7 reveals the constancy of certain agencies in causing the injuries.

The causes and means of prevention of accidents at California mines are discussed in the annual reports[12] of the United States Bureau of Mines and in several individual reports.[12] Probably no single item offers a more fertile field for reducing California mine accidents than the use of explosives, which accounts for 20 percent of the fatalities and coincidentally the same percentage of the accident cost. This item shows the significance of the human suffering and economic loss resulting from unsafe explosives practices.

Mine fires are more important[13] than is generally realized and are increasing as potential factors for disasters involving groups of men, owing to the increasing number of small-mine operations. During the 16-year period 1915-30, inclusive, 55 (10.1 percent) of the fatalities in California mines were caused by mine fires and 2 methane-gas explosions; from 1924-27, 2.5 percent of the fatalities were due to mine fires; from 1928-32, 4.5 percent; and during 1934 there were several fires, but fortunately no lives were lost.

A remedy for improving accident severity, which has shown almost a level trend for the past decade, is indicated by the causes of fatal and nonfatal accidents.

Fatalities in California for 1934 rank in order of importance and percentage of total as follows: (1) Falls of back or face and slides, 33.46 percent; (2) explosives, 17.69 percent; (3) falling in shafts, 14.63 percent; (4) falls of persons, 7.69 percent; (5) haulage, 4.61 percent; (6) falling and flying objects, 4.23 percent; (7) machinery, 3.85 percent; (8) mine fires and explosions other than explosives, 3.07 percent; (9) suffocations from

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12 Adams, W. W., Metal-Mine Accidents in the United States During the Calendar Year 1932: Bull. 377, Bureau of Mines, p. 5.
Accident-Prevention Record of the Metal-Mining Industry in the United States in 1933, Including Nonmetallic Mineral Mining Other Than Coal: Health and Safety Statistics 185, Demographical Division, Bureau of Mines, Dec. 6, 1934.
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Ash, S. H., Safety Practices in California Gold Dredging: Bull. 352, Bureau of Mines, 1932, 31 pp.

13 Glaeser, Oscar A., Accident-Prevention in Metal Mines: Min. Cong. Jour., December 1934, pp. 18-21.

mine gases and deficiency of oxygen, 2.31 percent; (10) others, including electrocutions and floods, 8.46 percent.

As regards relative hazardousness of location underground and man-hours of exposure in breaking ground, in which operation most fatal and nonfatal accidents occur, drifting is the least hazardous and sinking shafts, stoping, and driving raises, in order named, the most hazardous.

TABLE 8.- Rank and percentage of nonfatal accidents in California mines according to cause, 1924-34

Rank	Cause	Percentage
1	Falls or slides of rock or ore from back or wall..........	20.02
2	Timber or hand tools....................................	12.76
3	Haulage...	10.04
4	Falls, runs, or slides of rock or ore while loading at face, from chute, pocket or bins, and while shoveling or handling material at grizzlies......................	9.43
5	Falling down chute, winze, raise, or stope...............	8.17
6	Drilling..	7.20
7	Machinery...	4.83
8	Stepping on nails.......................................	3.64
9	Falls of persons..	3.34
10	Skips, cages, or buckets................................	1.79
11	Explosives...	1.20
12	Objects falling down shaft..............................	.49
13	Falling down shaft......................................	.40
14	Electricity...	.39
15	Suffocation from mine gases..............................	.18
16	Mine fires..	.10
17	Other causes..	16.02

According to table 8 certain causes of accidents form a high percentage numerically and in severity rates for fatal accidents, with a low percentage in frequency and number in nonfatal accidents; to this class belong accidents from skips, cages, or buckets, with explosives and others ranking 12 to 16. On the other hand, such accidents are decidedly severe and form the major items in permanent and partial disability claims when they do not result in fatalities. These items are of prime concern to the insurance carrier.

The importance from a compensation standpoint of some classes of injuries is not revealed by the cause; one nonfatal accident in a California mine resulting from the use of hand tools cost $15,228 for compensation and medical aid.

The parts of the body that are most difficult to protect in mine accidents are hands, insteps, and legs; the importance of such protection is emphasized by the fact that injuries to the hands ranked third in importance and constituted 10.23 percent of the injuries in 1,778 nonfatal accidents in California mines, while fingers ranked first with 18.24 percent and legs second with 10.62 percent.

The importance of protective clothing is shown by the classification (in the study of the 1,778 nonfatal accidents mentioned above) of injuries according to part of body injured, which ranked in importance and percentage as follows: (4) Feet, 10.18 percent; (6) eye, 9.40 percent; and (8) head, 5.62 percent.

In a study of 1,638 nonfatal and 42 fatal injuries at California metal mines the injuries according to nature ranked as follows: (1) Cuts, lacerations, and punctures, 30.16 percent; (2) bruises, contusions, and abrasions, 23.46 percent; (3) miscellaneous, 19.38 percent; (4) sprains and strains, 12.81 percent; (5) fractures, 9.4 percent; (6) burns and scalds, 1.79 percent; (7) dislocations, 0.78 percent; (8) loss or amputations, 0.78 percent; (9) stiffness, loss of function, 0.72 percent; (10) others, 0.72 percent.

It is observed that certain injuries of low frequency (8 to 10) have marked severity and high accident costs.

COST OF ACCIDENTS

The cost of accidents can be divided into indirect and direct costs. The indirect cost is variously estimated from 2 to 5 times the direct cost and is not enumerated in this report. The direct cost may be divided into loss of time, loss of wages, and industrial insurance cost, although usually it includes only compensation and medical and hospitalization expenses. The industrial insurance cost is variable; it covers the pure-premium cost and insurance overhead. The pure-premium cost includes compensation benefits and medical costs, items directly affected by the accident experience at each risk. This report deals primarily with the pure-premium cost of accidents; it does not discuss the factors[14] governing the establishment of compensation insurance rates.

Tables 10 to 13 have been compiled from data furnished by the California Inspection Rating Bureau and State Compensation Insurance Fund and cover mining risks insured by the insurance carriers in California for Class 1164, Mining, N.O.C.

A comprehensive study of accident costs in California reveals that the costs, as reflected by the insurance carriers' risks, can be applied with consistent results to all mining operations included in the Bureau of Mines classification and the accident experience for all mines discussed in this report, indicating that attention to safety is about the same in the majority of mines throughout the mining industry of the State.

The total benefit cost of injuries cannot be estimated from an analysis of certain high-severity injuries because permanent-disability awards are not

14 Fuller, G. V., Factors Governing the Establishment of Compensation Insurance Rates: Pit and Quarry, Dec. 30, 1931, pp. 58-62.

necessarily in direct ratio to time lost; a crippled man does not necessarily need medical relief indefinitely. Furthermore, although fatalities constitute the maximum lost-time figure affecting accident severity, nevertheless they rank with the lowest-cost injuries in California and at times do not rank in unit cost with even the average disability-cost figure; they vitally affect compensation costs but usually do not materially influence medical costs.

A study of 1,638 accidents, including 42 fatalities, at 355 representative California metal mines shows that 51.6 percent of the temporary-disability injuries are noncompensable -- or have a time loss of less than 8 days -- and that the average compensable temporary-disability injury has a time loss of 61 days. The periods of disability per claim for other injuries in this study were: Minor permanent partial disability, 240 days; major permanent partial disability, 1,135 days; average for all permanent partial disabilities, 474 days; permanent total disability, 2,749 days; and deaths, none. The average time lost for all injuries is 101 days, which compares favorably with the average days lost (109) per accident[15] for 1928.

The cost of accidents given in this report is based upon cost experience by policy years, which means that the experience for a policy year represents the experience developed 1 year from the date of the policy and can include not only the experience of the calendar year in which the policy was written but also experience developed during the life of the policy in the following calendar year. For this reason the experience shown in this report as 1932 policy year includes all experience developed during the calendar year 1933 of policies written during 1932.

Table 9 shows the benefit-cost experience of injuries in the metal-mining industry in California for the various classes of claims by policy years.

During the 5-year periods 1924-28 and 1928-32 the cost of compensation and medical aid increased, principally because permanent total disability and fatality claims increased both in unit cost and total costs. Although the total pay roll for 1928-32 was $6,255,417 less than for 1924-28, these classes of injuries increased both in total and in unit cost.

It is encouraging to note that a decided improvement is shown for 1932, but for the calendar year 1934 fatal-accident experience is worse than the average for the past decade.

The average benefit cost per accident for all insurance carriers' risks for the 5 years 1924-28 was $453; for the 5 years 1928-32, $560; and for 1932, $477.

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15 Adams, W. W., Metal-Mine Accidents in the United States During the Calendar Year 1928: Bull. 320, Bureau of Mines, 125 pp.

TABLE 9.- Accident-cost experience for Class 1164, Mining, covering California mines insured by insurance carriers for policy years 1928-32, inclusive, and summary for 1924-28 and 1928-32

Class of injury	1924-28	1928	1929	1930	1931	1932	1928-32
Temporary disability:							
Number........................	2,855	440	574	449	386	265	2,114
Compensation cost (total).....	$249,375	$43,674	$65,976	$44,836	$46,403	$23,752	$224,641
Compensation cost (per claim).	$87	$99	$115	$100	$120	$90	$106
Permanent partial disability:							
Number of minor claims........	149	30	38	14	30	13	125
Compensation cost (minor).....	$137,504	$25,859	$34,233	$13,358	$21,536	$13,497	$108,483
Compensation cost (per claim).	$923	$862	$902	$954	$718	$1,038	$868
Number of major claims........	84	12	12	8	16	10	58
Compensation cost (major).....	$271,262	$54,419	$34,265	$26,360	$45,188	$21,944	$182,176
Compensation cost (per claim).	$3,665	$4,535	$2,855	$3,295	$2,824	$2,194	$3,141
Number of P P D claims........	223	42	50	22	46	23	183
Compensation cost (P P D).....	$408,766	$80,278	$68,498	$39,718	$66,724	$35,441	$290,659
Compensation cost (per claim).	$1,832	$1,911	$1,370	$1,805	$1,450	$1,540	$1,588
Permanent total disability:							
Number........................	9	4	1	4	1	1	11
Compensation cost (total).....	$159,440	$75,748	$19,462	$68,497	$14,000	$22,249	$199,956
Compensation cost (per claim).	$17,715	$18,937	$19,462	$17,124	$14,000	$22,249	$ 18,178
Deaths:							
Number........................	96	13	30	21	14	7	85
Compensation cost (total).....	$130,666	$23,843	$71,993	$52,366	$39,514	$9,095	$196,811
Compensation cost (per claim).	$1,883	$1,834	$2,400	$2,493	$2,822	$1,299	$2,315
Grand total:							
Number........................	3,182	499	655	496	447	296	2,393
Compensation cost (total).....	$998,247	$223,543	$225,929	$205,417	$166,641	$90,537	$912,067
Medical cost (total)..........	$442,398	$91,782	$101,773	$91,023	$94,174	$50,472	$429,224
Benefit cost (total)..........	$1,440,645	$315,325	$327,702	$296,440	$260,815	$141,009	$1,341,291
Compensation cost (per claim).	$314	$448	$345	$414	$373	$306	$381
Medical cost (per claim)......	$139	$184	$155	$184	$210	$171	$179
Average benefit cost (per claim)	$453	$632	$500	$598	$583	$477	$560

Table 10 shows the annual pay roll for the mines, Class 1164, N.O.C., insured by the California insurance carriers, with the various items relating to accident costs per $100 of pay roll.

California compensation insurance rates are calculated to yield a ratio of pure-premium or benefit costs to earned premium of 59.4 percent. The loss ratio for the two 5-year periods 1924-28 and 1928-32 are coincidentally 69.4 percent, but 2 years (1927 with 51.7 percent and 1932 with 52.7 percent) have dropped below the authorized ratio.

The earned-premium cost per accident reached a high of $904 in 1932, and the average for the period 1928-32 was $808 compared with $653 for the 1924-28 period.

The benefit cost for 1932 of $4.14 per $100 of pay roll is the lowest in the last decade except for 1927, when it was $4.12. The compensation cost for the policy year 1932 of $2.66 per $100 of pay roll reflects the accident experience cost for the calendar year 1933 and is the lowest for this item in over a decade.

Table 11 gives number and percentages of cases in each class of injury with the cost for each class for the two 5-policy-year periods 1924-28 and 1928-32, including actual cost experience developed at the California insurance carriers' risks for the calendar years 1924-33, inclusive.

A detailed analysis of California's accident experience shows a decidedly level trend over the past decade. Data for individual years are not a satisfactory criterion under such circumstances; 5-year periods indicate more accurately the status of factors bearing on the occurrence and cost of accidents.

Compensation Costs

Although temporary-disability claims constituted 88.34 percent of all claims for the period 1928-32, they accounted for only 16.75 percent of the pure-premium cost -- a slight decrease in the number (89.69) and cost (17.31) for the period 1924-28. Although these figures show a decrease in the number of injuries in this class, the unit compensation cost of temporary disabilities was $106 for the 5-year period 1928-32 and $90 in 1932, compared with $87 for the 5-year period 1924-28. Unfortunately, this type of claim causes the most irritation in accident-cost discussion. In this class malingering assumes large proportions, whereas in reality the cost of these claims is comparatively low when compared with that of permanent disability and (to some extent) fatalities. Although during the policy year 1932 there was only 1 permanent-total-disability claim, the cost in compensation was only $1,503 less than that for the entire 265 temporary-disability claims (table 9), and the 23 permanent-partial-disability claims cost $11,689 more in compensation than the entire 265 temporary-disability claims, or 49.2 percent of the total compensation cost of these claims. Also, during the 5-year period 1928-32, 85 permanent-disability claims cost $40,516 more than 96 similar claims for the 5-year period 1924-28. There is no doubt as to the seriousness of permanent-disability injuries

TABLE 10.- Industrial insurance costs per $100 of pay roll per accident; earned premium-loss ratio; pay roll covering California mines, Class 1164, insured by insurance carriers for the policy years 1928-32, inclusive, and summary for years 1924-28 and 1928-32

Item	Year (Policy)						
	1924-28	1928	1929	1930	1931	1932	1928-32
Pay roll	$28,309,565	$4,591,711	$5,566,962	$4,630,245	$3,868,528	$3,396,702	$22,054,148
Earned premium	$2,078,047	$392,862	$515,131	$373,382	$382,594	$267,791	$1,931,760
Rate per $100 of pay roll (manual)	------------	$8.53	$9.05	$9.42	$10.54	$10.99	------------
Costs per $100 of pay roll:							
Compensation	$3.52	$4.87	$4.06	$4.43	$4.30	$2.66	$4.14
Medical	$1.56	$2.00	$1.83	$1.97	$2.43	$1.48	$1.95
Total benefit costs	$5.08	$6.87	$5.89	$6.40	$6.73	$4.14	$6.09
Earned premium	$7.44	$8.56	$9.25	$8.06	$9.87	$7.87	$8.76
Loss ratio (percent)	69.4	80.3	63.6	79.3	68.3	52.7	69.40
Industrial insurance[1] cost per claim	$653	$787	$786	$753	$857	$904	$808

1 Represents benefits and insurance administration costs.

TABLE 11.- Review of injuries showing percentage of total number and percentage of total compensation and medical costs by class of injury, Class 1164, Mining, 1924-28 and 1928-32 1

Class of injury	Percentage of total number of injuries		Percentage of total cost of injuries	
	1924-28	1928-32	1924-28	1928-32
Temporary disability:				
Noncompensable:				
Medical cases..................	-------	45.58	-------	2.42
Compensable:				
Compensation..................	-------	-------	17.31	16.75
Medical........................	-------	-------	-------	16.07
Subtotal......................	-------	42.76	-------	32.82
Total............................	89.69	88.34	-------	35.24
Permanent partial disability:				
Minor:				
Compensation..................	-------	-------	9.54	8.08
Medical........................	-------	-------	-------	3.36
Subtotal......................	4.68	5.22	-------	11.44
Major:				
Compensation..................	-------	-------	18.81	13.58
Medical........................	-------	-------	-------	5.66
Subtotal......................	2.35	2.43	-------	19.24
Total:				
Compensation..................	-------	-------	28.35	21.66
Medical........................	-------	-------	-------	9.02
Subtotal......................	7.03	7.65	-------	30.68
Permanent total disability:				
Compensation..................	-------	-------	11.07	14.91
Medical........................	-------	-------	-------	4.08
Subtotal......................	.28	.46	-------	18.99
Deaths:				
Compensation..................	-------	-------	12.57	14.67
Medical........................	-------	-------	-------	.42
Subtotal......................	3.00	3.55	-------	15.09
All injuries:				
Compensation..................	-------	-------	69.30	67.99
Medical........................	-------	-------	30.70	32.01
Grand total......................	100.00	100.00	100.00	100.00

1 Policy years.

which result in loss of limbs or eyes or broken backs; to eliminate them the prevention of accidents from falls, explosives, haulage, and falling down shafts should receive more attention, and first aid should be applied at more small mines to prevent infection and other complications from both major and minor injuries.

Permanent-partial-disability claims constituted 7.65 percent of the claims for 1928-32 compared with 7.03 percent for 1924-28. On the other hand, the compensation cost of these claims compared with the total decreased from 28.35 to 21.66 percent. The unit cost per accident of this type shows a steady decrease, due in part at least to first-aid education which the industry at the larger mines has undertaken energetically since 1930.

Medical Costs

Medical costs (table 11) for the 5 years 1928-32 were 32.01 percent of the total pure-premium cost and in 1932, 35.8 percent, compared with 30.7 percent for the 5 years 1924-28. Medical care for temporary-disability injuries accounted for 18.49 percent of the total pure-premium cost and approximately 58 percent of the total medical cost. Medical cost for permanent partial disabilities accounted for 9.02 percent; permanent total disabilities, 4.08 percent; and fatal injuries, 0.42 percent of the pure-premium costs.

Medical costs are controlled largely by the procedure followed in temporary-disability injuries. The small mine plays a more important part than is realized. Otherwise than by reducing accidents, these costs can be controlled effectively only by getting the injured men back to work through the efforts of the insurance department, the doctor, and the employers. These three agencies should be advised of the status of any workmen receiving medical care.

COST OF ACCIDENTS BY CAUSES

Table 12 reviews accident costs by causes, and table 13 ranks injuries by causes in the order of importance indicated by cost, percentage of total number of injuries, and severity (time lost) of injuries.

Prevention of accidents by causes has been discussed in this report. The necessity for safer mining practice at the face is emphasized by the fact that falling objects account for 34.21 percent of the total cost of accidents and rank first in cost, number of injuries, and severity (direct measure of human suffering).

The misuse of explosives accounts for 21.46 percent of accident costs in California, ranking second in cost and severity and ninth or last in frequency; this latter fact is very misleading regarding explosives accidents in California. No single item in California mining could contribute more to accident prevention than a drastic change towards safer explosives practice. The average nonfatal injury resulting from the misuse of explosives cost $6,400. The maximum unit cost of $5,115 per injury is accounted for by explosives; injury from explosives costs 14½ times as much as the average injury and approximately 6 times the unit cost ($872) for haulage injuries, which rank second in unit cost.

TABLE 12.- Review of accident-cost experience, latest available period,[1] by cause of injury

Cause and class of injury [2]	Percentage of injuries	Relative severity[3]	Percentage of total cost of all injuries	Pure-premium cost per injury		
				Compensation	Medical aid	Total
Falling objects:						
In mines:						
Deaths	1.31	38.52	5.17	$1,333	$67	$1,400
All injuries	9.29	45.14	28.77	834	265	1,099
Collapse:						
All injuries	14.46	1.24	3.10	42	34	76
All others:						
Deaths	.06	1.75	.22	1,230	60	1,290
All injuries	3.63	2.33	2.34	134	94	228
Subtotal:						
Deaths	1.37	40.27	5.39	1,328	67	1,395
All injuries	27.38	48.71	34.21	323	120	443
Explosives:						
Deaths	.42	12.23	2.13	1,793	18	1,811
All injuries	1.49	16.53	21.46	4,200	915	5,115
Haulage:						
Mine cars and trucks on rails:						
Deaths	.06	1.75	.81	4,818	7	4,825
All injuries	1.73	2.45	3.19	435	221	656
Mine cages, skips, and buckets:						
Deaths	.12	3.50	.25	690	72	762
All injuries	.53	4.49	2.96	1,226	733	1,959
Automobiles:						
Deaths	.06	1.75	.03	150	15	165
All injuries	1.19	2.68	2.33	565	128	693
Subtotal:						
Deaths	.24	7.00	1.09	1,587	41	1,628
All injuries	3.45	9.62	8.48	603	269	872

(See page 30 for footnotes).

TABLE 12.- Review of accident-cost experience, latest available period,[1] by cause of injury - Continued

Cause and class of injury[2]	Percentage of injuries	Relative severity[3]	Percentage of total cost of all injuries	Pure-premium cost per injury		
				Compensation	Medical aid	Total
Handling materials:						
All injuries	23.39	2.66	7.21	63	46	109
Hand tools:						
All injuries	7.75	2.72	7.06	255	69	324
Falls of persons:						
From elevations and into excavations:						
Deaths	.12	3.50	.72	2,150	9	2,159
All injuries	4.35	4.69	4.52	245	123	368
On level:						
All injuries	5.54	.43	1.33	42	43	85
Subtotal:						
Deaths	.12	3.50	.72	2,150	9	2,159
All injuries	9.89	5.12	5.84	131	78	209
Stepping or striking against objects:						
All injuries	9.22	.97	2.66	57	45	102
Flying particles:						
All injuries	6.60	.50	1.28	36	33	69
Others (not segregated):						
Deaths	.35	10.50	4.74	4,681	32	4,713
All injuries	10.83	13.17	11.80	333	54	387
Grand total:						
Deaths	2.50	73.50	14.07	1,957	50	1,997
All injuries	100.00	100.00	100.00	263	92	355

1 Data compiled from experience of State Compensation Insurance Fund - 355 mines.
2 The item "all injuries" under each heading includes deaths, if any.
3 Deaths charged 6,000 days each, nonfatal injuries actual lost time awarded.

TABLE 13.- Causes of injuries ranked in importance according to cost, percentage, number, and severity

Cause	Rank according to -		
	Percentage of total cost of all injuries	Percentage of total number of injuries	Severity of all injuries (time lost)
Falling objects	1	1	1
Explosives	2	9	2
Cause not segregated	3	3	3
Haulage	4	8	4
Handling materials	5	2	7
Hand tools	6	6	6
Falls of persons	7	4	5
Stepping or striking against objects	8	5	8
Flying particles	9	7	9

CONCLUSIONS

A frequent complaint in various States is that compensation costs are high and are increasing. Compensation-insurance-cost rates have risen consistently, owing not so much to increased administration costs as to various other factors. The sooner it is recognized that accidents are preventable, (and in many instances actually have been prevented) the sooner they will be prevented and accident costs decline, regardless of other factors that tend to push the rates upward. In other words, the remedy for the high cost of accidents in practically all of its phases is the prevention of their occurrence, and this remedy is available to the mining industry.

A study of accident costs[16] in other Pacific Coast States shows no noticeably wide divergence in the costs and classification of injuries in States where economic conditions are essentially similar; temporary-disability injuries in mining account for 88 percent of the total number in Arizona and 88.1 percent in Washington, compared with 88.34 percent in California.

The increasing number of prospects and small "partnership" operations has injected in the California mining industry factors of growing importance as contributors to accidents, as lack of supervision, unsafe practices, unsafe equipment, and other hazards. Indirectly if not directly these will contribute much toward high accident rates with increased costs to the industry unless more attention to accident-prevention work is required in and around these small

16 Ash, S. H., Accident Experience and Cost of Accidents at Washington Metal Mines and Quarries: Tech. Paper 514, Bureau of Mines, 1932, 35 pp.

mines. Under the compensation laws of California partners who employ no workmen are classed as owners and do not come under the safety provisions or compensation insurance provisions of the compensation act; unquestionably this classification has a definite tendency to encourage laxity in safety in the operation of these properties.

According to statistics in this report, on the average 22 persons are killed annually at California mines, and for every person killed 60 others are injured. As a result of these nonfatal injuries approximately 60,000 days of wages are lost to the workmen aside from reduced earning capacity due to physical impairment. The wages, which alone amount to at least $270,000 annually, are returned only in part through compensation insurance. The industry is now paying annually about $1,074,000 for industrial awards -- a tax each year of at least 10 percent of California's annual gold production which gives no return except human wreckage and dependency in the community and wasted dividends. It can be reduced very materially by an increased effort to prevent accidents, possibly only by concerted continued effort, chiefly of an educational nature, but with the application of pressure by the State on those who refuse or fail to cooperate.

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INFORMATION CIRCULAR

SOME OBSERVATIONS AS TO SAFETY HAZARDS IN 47 NORTHERN COLORADO SUBBITUMINOUS COAL MINES

BY

E. H. Denny

I.C. 6862,
November 1935.

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

SOME OBSERVATIONS AS TO SAFETY HAZARDS IN 47 NORTHERN COLORADO SUBBITUMINOUS COAL MINES[1]

By E. H. Denny[2]

During 1934 the United States Bureau of Mines through the Denver office of the Safety Division cooperated with N.R.A. Divisional Coal Code Authority Division V, District 1 (Northern Colorado producers) in sampling 47 northern Colorado subbituminous coal mines; the work presented an excellent opportunity to observe operating and safety conditions and practices in this group of relatively small mines. This paper deals chiefly with safety conditions. The sampling was done to assist the code authorities in the classification of coals; analyses of the samples were made by the coal laboratory of the Bureau at the Pittsburgh (Pa.) Experiment Station and are appended hereto.

NORTHERN COLORADO COALS

The term "northern Colorado coals" used in this report refers to coal mined in Boulder, Weld, Jefferson, El Paso, Jackson, Larimer, and Elbert Counties, Colo., all of which is subbituminous and relatively high in moisture content. In 1933 the total coal production[3] of the counties listed was 2,417,757 tons.

The coal produced is consumed largely in Denver and adjoining counties and by the railroads of the district. The distance to which it can be shipped out of the State is limited to some extent by the ease with which it slacks and fires on exposure to air.

All the larger producers except one are shaft mines, the shafts ranging up to 788 feet in depth. Coal is mined by the room-and-pillar system, and pillars are largely extracted.

The coal is cut by machines and sometimes sheared as well. Mechanical loading is just being introduced in the field. Where the thickness of the coal permits, 16 or more inches of roof coal are left for roof support, as the main roof is usually poor-quality shale.

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1 The Bureau of Mines will welcome reprinting of this information circular, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6862."

2 District engineer, Safety Division, Health and Safety Branch, U.S. Bureau of Mines, Denver, Colo.

3 Report of Colorado State Coal-Mine Inspector.

The coal beds in the larger mines usually are nearly flat with local irregularities; the western part of the field pitches sharply and is almost vertical in a few openings. Faulting is extensive, and sometimes the displacement prevents extension of the mine workings. The geology of the field is described in publications[4] of the United States Geological Survey. Mining practices of some of the larger mines are described in Bureau of Mines Report of Investigations 3199.[5]

OBSERVATIONS ON HAZARDS IN MINING OPERATIONS

Underground and surface sampling of the 47 mines presented an opportunity to observe and inquire into mining conditions with particular reference to safety. Excellent safety practices were in vogue at many of the mines; good practices, however, are not discussed in this report. In the smaller "wagon" mines safety conditions and practices varied widely. Conditions naturally were better in those small operations manned by experienced miners. A number of such mines have very capable and experienced officials and miners; many of them, however, are operated with scanty equipment and little capital under adverse natural mine conditions by men striving to avoid relief rolls, who expose themselves and their fellow workers to considerably more hazards than would confront them in well-planned and equipped mining operations. Some of these hazards can be minimized readily; others can be abated or eliminated only by considerable expenditure. It seems questionable whether men for the sake of a meager living should risk or be allowed to risk life and limb and the welfare of their dependents without the ordinary safeguards commonly accepted in most mines through experience.

The substandard or unsafe conditions or practices detailed below were observed in one or more mines. All these conditions and practices do not obtain in any one mine; most of them prevail in the smaller mines and are sufficiently widespread to warrant the conclusion that much improvement is desirable in the smaller mines as a class. A few notes apply to mines in the Southern Colorado field visited during the year. Much more time and opportunity were available in the smaller than in the larger mines to examine and observe conditions while sampling.

Roof Hazards

Falls of roof and face account for about half of the coal-mine fatalities underground and a large proportion of the broken backs and other serious injuries received in all coal mines. Most of these could be prevented by adequate

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4 Emmons, S. F., Cross, Whitman, and Eldridge, G. R., Geology of the Denver Basin in Colorado: Mon. U.S. Geol. Survey, vol. 27, 1896.
Martin, G. C., Coal of the Denver Basin; Coal Fields in Colorado and New Mexico: U.S. Geol. Survey Bull. 381c, 1908, pp. 297-306.

5 Tomlinson, H., A Study of Falls of Roof and Coal in Northern Colorado: Rept. of Investigations 3199, Bureau of Mines, 1933, 20 pp.

timbering placed when and where needed and by barring down loose and drummy material. The following are extracts from notes on individual mines:

Timbering throughout the parts of the mine traveled was inadequate and was set in a makeshift manner.

Roof conditions appeared to be dangerous in several working places, particularly where the immediate roof was shale.

Overhanging draw slate is a hazard and should be cleaned down more carefully.

For the limited amount of development roof conditions were particularly bad. The roof of one main entry contained a large piece of loose roof coal weighing half a ton or more; the other entry roof contained a piece of draw slate about 18 inches by 5 feet by 10 feet which had partly separated about 4 inches from the sandy shale roof. The oldtime miners working nearby and tramming under the loose slate planned to take it down when they had time and inclination.

In a mine employing normally about 100 men "falls had occurred in 3 rooms over Sunday spanning the entire roadway and extending 15 to 20 feet in length." Crossbar sets should be placed at close intervals.

In collecting samples at a small operation the author had to crawl over 5 major falls in the main entry, and during the sampling 3 or 4 tons of roof fell near by, knocking out 2 sets of timber. Anything is used for timber from old ties to cottonwood. No roadhead props are used. The roof is a poor spongy slate which slacks readily and should be timbered systematically. However, the operators prefer to timber scantily and spend considerable time cribbing in order to continue operations.

In various places throughout the mine dangerous roof conditions caused the three men working underground no concern.

The timbering agreement with the State mine inspector calls for timber on 5-foot centers. A number of places do not comply with this, and more timbering is needed than that required by the agreement. Serious roof-fall accidents occurred recently in this large mine.

Haulage Accidents

Haulage accidents result in many permanent partial disabilities and fatalities.

The following notes are abstracted from reports of haulage conditions observed in small mines each having one or more of the following unsafe practices or conditions:

In one mine a surface derail switch on the slope is spiked closed. The main-slope man-clearance is insufficient, and no manway is provided.

Mine cars are badly in need of repair. No derailing device is used at the top of the main slope.

The one mine car used for haulage lacks half of a bottom board and an entire end board.

There is no track in the mine; a box with sheet-metal bottom holding about half a ton of coal is pulled to the surface by an automobile engine and cable.

The hoist rope passes from the hoist by a series of sheaves along the ground to the foot of the tipple with some hazard to passersby. No drag is used on the slope, and there is no derail. Switches are of the foot-throw type.

Bony coal and rock gobbed along the slope leave insufficient man-clearance. The hoist landing indicator was not in use, the position of the trip being gaged by the turns of rope on the drum.

The hoist cable is in very poor condition and is also badly in need of greasing. Broken strands are frequent throughout the cable length.

The bell system is a jerk-line from the bottom of the shaft to a bell at the shaft mouth. The signals heard at the hoist 75 feet away were easily mistaken. The indicator on the hoist probably will be put in working order.

The skip cable showed considerable wear from nonuse of rollers or sheaves on the slope.

Cars are not in good repair, and the track is rough and uneven.

During hoisting on the slope the rope whips about, involving hazard to persons traveling the slope, and the man-clearance on the slope is limited.

Safety catches are not installed on the cage, and overwind devices are not used. An improvised bell system is usually supplanted by shouting from the shaft bottom. Six-pound rails are used.

Hoisting from the shaft is by a horse-drawn windlass without a brake. The harness on the horse is old and in poor condition. Cages are counterbalanced. Underground tramming is done by hand, and as the low entries are not brushed men crouch on their knees

while pushing cars. Most of the track in the mine is of 2-inch by 4-inch boards. No room switches are used, cars being lifted off the entry track and turned onto the room tracks.

Insufficient man clearance was noted between the loaded track and the rib near the shaft bottom at this large mining operation.

At one medium-size operation the floor in parts of the mine was heaving, and in places one side of the track was a foot higher than the other side. The cars were not in good repair, and spillage of coal was excessive.

At another medium-size operation the track was crooked and rough. The haulageway was badly "cluttered up" with coal and rock which made walking difficult. Many of the cars were in need of repair. Considering the condition of the track, the man-trip should travel more slowly. Cagers wisely keep their distance during coal hoisting because of coal spillage down the shaft.

In one large mine miners top their cars excessively, with the result that a great deal of coal is spilled along haulageways.

Over 600 feet of the roadbed was ankle deep in mud, and in some places the rails were under water. A number of falls were noted along haulageways, and timbering was done in makeshift fashion. In several places the crossbars had taken so much weight that the mule had to step outside of the track in order to pass under. Several broken strands were noted in the slope hoist rope.

Five of the smaller mines visited used automobile motors for hoisting coal, and one used a steam-operated thrashing tractor for operating a hoist and compressor. Considerable ingenuity was manifested in the adaptation of these motors to haulage of coal. Most of them handle one 1-ton car of coal. Usually these motors are not suitable because of insufficient speed control and lack of overwind device. In one instance the automobile motor drove a drum of a hoist which was equipped with a hand brake; and a brake lever extended from the automobile motor-drive shaft to the hoist operator's position.

Ventilation and Gases

Methane gas has been encountered occasionally in some of the larger mines of the northern Colorado field, although detection of methane is rather uncommon. It has been ignited on several occasions, burning men. Closed lights are used in most of the larger mines with "no-smoking" rules, and fireboss inspections are required. Carbon dioxide and nitrogen mixtures (black damp) are much more common than methane. Black damp is formed quite readily by absorption of oxygen or oxidation of subbituminous coal. The larger mines are ventilated by relatively large-capacity fans, but in some of these the circu-

lation of air in room entries is noticeably sluggish because too much reliance is placed on leaky doors in slants, and canvas brattice is used too freely and installed loosely in numerous room and entry crosscuts.

The following notes illustrate common occurrences of black damp or carbon dioxide when ventilation is inadequate or improperly directed.

> In two working faces in small mines black damp was found in percentages high enough to affect a carbide lamp. It is understood that the fan is shut down after operating about an hour each morning.
>
> In most of the faces visited the ventilation was very poor, and the black damp present affected the carbide lamps. An attempt to inspect the sump failed because the superintendent's carbide lamp would not burn due to black damp present.
>
> Whenever the barometer drops, enough black damp is forced out of the old workings to extinguish the workers' carbide lamps.
>
> Recementing of cracks in stoppings is frequently necessary to prevent the admission of fumes from the fire to the main slope.

In a number of the smaller mines substandard ventilation practice is common; 8 small northern Colorado coal mines and 2 small coal mines in other parts of the State relied solely upon natural ventilation. It is true that these mines were operating only a short distance from surface openings, but unquestionably conditions in these mines are such that good circulation of air induced mechanically is essential for safety against black-damp accumulations. In the event of fire natural ventilation is likely to reverse and imperil men, as illustrated by the loss of a number of lives in northern Colorado; a mechanically induced air current can be utilized in directing the air in fighting fire and sealing off fire areas and offers a better chance for the men to escape than natural ventilation. In several mines the fan was said to be operated intermittently or shut down at night to save power costs; this practice permits gases from old workings or sealed-off fires to impede the work of the mine and endanger the men. One mine had a number of rooms turned but had only one opening, a shaft; it was ventilated by blower and tubing. Two instances of considerable air leakage were noted at the fan from leaky fan housing; much of the air intended for the mine was short-circuiting at the fan.

Fire Hazard

Reference has been made to the generally recognized hazard of spontaneous ignition of subbituminous coal. Nevertheless, most of the mines, both large and small, lack underground water lines to or near the faces for prompt combatting of fires. So far as known only one northern Colorado coal company owns modern oxygen breathing-apparatus equipment, and this was not being maintained

ready for use. The United States Bureau of Mines maintains oxygen apparatus and gas masks on a truck and temporarily on a railroad mine rescue car at Denver; this apparatus is used frequently in remote parts of Colorado or outside the State. Training of northern Colorado coal miners in the use of this equipment had been almost completely abandoned owing to company economies and the Bureau's lack of funds for this work, but in the spring of 1935 some training was conducted through the cooperation of mining companies, the State of Colorado, and the United States Bureau of Mines. Fortunately most of the many fires occurring in subbituminous mines develop slowly, and heated areas and fire areas are usually sealed without great difficulty.

In pitching coal operations it was observed that much coal was left on the hanging and foot walls. Later falls of coal leave fine coal in the room necks which is likely to ignite; sometimes not enough thought is given to laying out workings to facilitate sealing of such likely fire areas. In some operations considerable gobbing of fine coal in rooms adds to the danger of spontaneous ignition.

The natural fire hazard is frequently enhanced by nonfireproof construction and poor housekeeping. Some wooden tipples had poorly hung, open electric wiring and little or no fire protection. Spillage of oil and grease and accumulation of combustible debris are common in and about these tipples. In one instance hot ashes were used for the tipple foundation, and the waste dump was burning within 25 feet of the wooden tipple. The wooden joists of the fan building were largely soaked with oil. Surface buildings of timber construction are sometimes located so as to form a fire hazard to men underground. In some instances wooden fan buildings and connections to airways present a hazard in that fumes might be drawn into the mine from a surface fire at or around this fan. Several dry timbered hoisting shafts were adjacent to wooden surface structures; in one mine such a shaft was the only exit.

The common use of granular black blasting powder and pellet powder in the field also introduces an additional hazard in firing coal. At least one recent fire is thought to have started from a shot of pellet powder; unquestionably numerous fires are caused by this unsafe method of coal-mine blasting.

Many companies keep fire extinguishers in surface buildings, but some of these extinguishers are not types approved by the Fire Underwriters and probably would be effective only in incipient fires.

Surface fire protection ranges from numerous hydrants and hose and good water pressure at some mines to no fire protection at many of the smaller properties.

In two small mines hay was scattered in and about the underground stable and the adjacent entry, constituting a fire hazard, and in one large mine apparently excess hay was stored underground.

Explosives

This work afforded little opportunity to observe explosives practice. Pellet powder, granular black blasting powder, and permissible explosives are used in the mines. Shooting commonly is done at the close of the shift or at noon (or both) by shot firers or miners. In general, coal is cut and also sheared; some solid shooting is practiced in the smaller mines.

At one small mine gelatin dynamite, detonators, and pellet powder were stored together in a surface magazine. In two mines explosives and detonators were kept together in a powder box. In another prepared primers and 20 sticks of explosives were in a box on the entry close to the track.

At two of the larger mines holes had been loaded before the place was undercut, and during the slack season loaded holes are sometimes not fired for several days. Such practices result in blown-out shots, possible explosions, and misfires and the numerous hazards accompanying their handling.

Several large and small mines used coal-dust drillings and slack coal for stemming holes.

Transportation of explosives in a second trip closely following the man-trip was reported in one large mine. In a small mine unboxed pellet powder and fuse were transported on a cage carrying 4 men wearing burning carbide lamps.

Dust

The haulage roads, manways, and air courses of several large and small mines were very dusty in places. Inasmuch as heavy blasting with granular black blasting powder has been practiced in northern Colorado for many years without reported instances of dust ignition mine operators consider that there is little or no danger of dust explosions. Tests of Wyoming subbituminous coal by the Bureau of Mines have shown that fine coal dust can be ignited by a blown-out shot of black blasting powder and produce a general explosion. More study is needed to determine what factors are involved in the apparent immunity from dust explosions of the northern Colorado subbituminous mines; meanwhile much more caution should be exercised than at present in the use of explosives in these mines. If a dust explosion should occur in a high-moisture coal of northern Colorado it probably would be started by the ignition of a considerable body of gas or by an electric arc in the presence of a large dust cloud from a wrecked trip or from the ignition of a considerable amount of explosives. Dust accumulations should not be disregarded but should be brushed or washed from ribs and timbers, and roadways should be cleaned regularly and as completely as practicable. Rock-dusting of these mines, especially those that are definitely dry, is desirable but is not likely to be practiced until after an explosion occurs.

Dust accumulations were noted in tipples, and two instances of spontaneous ignition of fine dust in a tipple were reported.

Electricity

Power for trolley locomotives and coal-cutting machines is commonly 250 and 440 volts potential. In some mines the trolley wires are well-guarded at places where men have to pass under them, as at branch entries and shaft bottoms; but even at such locations guards are often lacking, with consequent danger of electrocuting men. So long as men are exposed to unguarded trolley wires or power lines occasional loss of life is unavoidable. Some metal mines as well as a few coal mines have found that boxing the trolley wire throughout the mine is feasible and that the added protection to life pays in dollars and cents and in the prevention of suffering and misery.

In both large and small mines instances of substandard wiring on the tipple and underground were reported; substandard practices include wires touching timbers and coal, wires poorly supported, wires hung on nails or wooden wedges or wrapped around posts instead of on insulators, bare spots for connection to machine cables, and poorly insulated splices in cables and wires, often where clearance is limited and lighting inadequate. Such wiring is common in and about many coal mines of the United States although it would not be tolerated in industrial plants or city residences and should not be allowed in any kind of mine.

Miscellaneous

Some of the larger mines apparently lacked adequate supervisory force during the depression; at some of the smaller mines the underground foreman performed many duties underground, on the surface, and in town. Even with the present supervisory force, much more attention could and should be given to safety practices.

In some mines protective hats are used by all; in others there are none. In some mines goggles are commonly used in picking operations; at others where the miners are supposed to use goggles few or none were noted. Protective shoes are used only to a limited extent.

No check-in-and-out system is reported at one small mine.

First-aid material is usually available at least in the mine office, but one small mine had none.

At least two of the small mines are said to have no maps of the underground workings.

Poor plant maintenance was evident at several large and small mines.

Many surface and underground machines were unguarded or improperly guarded.

One underground-to-surface telephone system was not working, and a fire-boss vacancy at this mine had not been filled.

Inquiry at some small mines employing more than 4 men indicated that no compensation was carried.

In numerous instances substandard conditions and practices have been condemned by State mine inspectors and orders issued for their rectification, but the limited travel funds of the State department have made the prompt carrying out of such orders difficult or impossible. For mines that definitely come under the certified-foreman provision of the mining law (mines employing three or more men underground) the State has been fairly successful in obtaining a responsible supervisor in most cases.

CONCLUSION

As previously stated, the unsafe conditions and practices discussed herein do not obtain at all mines visited. Some small and some large mines have excellent safety conditions and practices. Good conditions and practices should be more universal, and conditions could be improved in many particulars with little present and probably no ultimate cost. Perhaps mines where capable men are employed with limited and poor equipment and working under adverse natural conditions could be combined to make better mines. It certainly does not benefit the public welfare for mining operations to expose men constantly to unnecessary hazards, with resulting loss of life or injury to be paid for ultimately by the State and its citizens.

The appended tabulations summarize the coal analyses made in connection with the sampling in this study; the analytical work was done by the coal laboratory at the Pittsburgh (Pa.) Experiment Station of the United States Bureau of Mines. Table 1 summarizes the data on face samples taken in the mines, and table 2 gives similar data from tipple samples, both the face sampling and the tipple sampling being according to Bureau of Mines standard methods.

TABLE 1.- Analyses of Mine Samples, Northern Colorado Coals

Where mined			Number of samples mixed and analyzed	Date sampled	Moisture, as received	Dry coal								B.t.u., as received	B.t.u., dry coal	B.t.u., ash- and moisture-free	Softening temperature, °F.
County and town	Mine	Bed				Volatile matter	Fixed carbon	Ash	Sulphur	Hydrogen	Carbon	Nitrogen	Oxygen				
Boulder County																	
Erie........	Arrow (Star).	Unnamed.	2	4/5/34	23.8	36.3	57.4	6.3	.6	4.7	72.1	1.7	14.6	9500	12460	13300	2305
Lafayette...	Black Diamond.	do.	4	3/26/34	21.5	38.0	56.3	5.7	.5	4.8	72.0	1.6	15.4	9790	12470	13220	2100
Gorham......	Crown....	do.	3	3/29/34	18.3	39.0	55.9	5.1	.4	4.9	71.8	1.6	16.2	10130	12400	13060	2175
Do.	Eldorado.	do.	3	3/12/34	19.1	39.0	55.5	5.5	2.0	4.8	70.2	1.1	16.4	9950	12300	13010	2120
Louisville..	Fireside.	do.	3	3/20/34	21.2	36.0	57.0	7.0	.6	4.7	71.4	1.4	14.9	9740	12350	13280	2100
Superior....	Gorham...	Gorham..	2	3/8/34	19.7	38.6	56.2	5.2	.4	4.6	71.9	1.5	16.4	9970	12420	13100	2145
Lafayette...	Hiway....	Boulder.	3	11/5/30	19.9	38.9	56.2	4.9	.4	4.9	73.5	1.6	14.7	10220	12760	13410	2175
Superior....	Industrial.	Unnamed.	4	11/3/32	18.9	37.4	57.1	5.5	.3	4.9	72.8	1.5	15.0	10110	12460	13190	2160
Gorham......	Lewis No. 2.	do.	3	3/19/34	18.4	39.6	54.6	5.8	1.5	5.1	70.6	1.2	15.8	10260	12560	13340	2230
Broomfield..	Monarch No. 2.	do.	4	6/13/34	19.6	38.0	57.2	4.8	.4	4.9	72.1	1.6	16.2	10130	12600	13240	2210
Lafayette...	Paramount	do.	2	3/23/34	20.5	38.9	54.3	6.8	.8	4.8	70.5	1.4	15.7	9780	12310	13210	2130
Gorham......	Pine Cliff.	do.	2	3/15/34	18.7	37.6	55.1	7.3	1.1	4.7	69.6	1.2	16.1	9890	12170	13130	2100
Superior....	Pluto....	do.	2	3/6/34	19.3	37.5	56.7	5.8	.5	4.7	72.2	1.5	15.3	10020	12430	13190	2120
Gorham......	Red Ash..	do.	2	3/29/34	18.9	37.4	56.1	6.5	1.2	4.7	70.5	1.2	15.9	9910	12220	13060	2090
Do.	Rosser...	do.	2	3/16/34	19.8	39.0	56.0	5.0	1.1	4.8	71.9	1.1	16.1	9990	12450	13100	2230
Elbert County																	
Matheson....	White Ash	Unnamed.	2	7/12/34	33.8	41.6	44.8	13.6	1.3	4.5	63.6	1.2	15.8	7220	10910	12630	2270

TABLE 1.- Analyses of Mine Samples, Northern Colorado Coals - Continued

Where mined			Number of samples mixed and analyzed	Date sampled	Moisture, as received	Dry coal								B.t.u., as received	B.t.u., dry coal	B.t.u., ash- and moisture-free	Softening temperature, °F.
County and town	Mine	Bed				Volatile matter	Fixed carbon	Ash	Sulphur	Hydrogen	Carbon	Nitrogen	Oxygen				
El Paso County																	
Colorado Springs.	Altitude.	A seam..	2	7/10/34	25.6	41.9	51.0	7.1	.5	4.6	67.9	1.0	18.9	8540	11480	12360	2395
Do.	City.....	Fox Hill	2	7/5/34	24.5	41.5	51.0	7.5	.5	4.7	68.1	1.0	18.2	8750	11590	12530	2330
Do.	Dixie....	A seam..	2	7/11/34	23.3	41.2	52.2	6.6	.4	5.0	70.4	1.2	16.4	9350	12190	13050	2445
Do.	Jimmy Camp.	do.	2	7/11/34	23.0	40.7	50.5	8.8	.6	4.8	68.6	1.2	16.0	9100	11820	12950	2220
Do.	New Keystone.	Unnamed.	2	7/10/34	24.7	41.8	50.8	7.4	.5	4.7	68.8	1.0	17.6	8810	11700	12640	2335
Pikeview....	Pikeview.	Fox Hill	7	6/28/34	25.8	41.9	51.8	6.3	.4	4.6	67.4	.9	20.4	8580	11550	12320	2380
Jackson County																	
Walden......	Marr.....	Sudduth.	2	7/20/34	15.9	39.6	56.8	3.6	.1	5.0	74.8	1.0	15.5	10370	12940	13420	2290
Coalmont....	Moore....	Riaach..	3	7/19/34	17.8	38.8	50.0	11.2	.4	4.8	65.8	1.9	15.9	9430	11460	12910	2605
Jefferson County																	
Leyden......	Fruth....	Unnamed.	2	6/22/34	25.5	42.1	47.9	10.0	2.0	4.7	64.2	.9	18.2	8340	11190	12430	2085
Do.	Leyden No. 3.	do.	4	5/10/34	22.0	37.8	56.5	5.7	.6	4.5	70.8	1.1	17.3	9380	12030	12760	2210
Littleton...	Unity....	do.	2	4/25/34	29.1	39.6	51.9	8.5	.6	4.4	67.4	1.1	18.0	8110	11430	12500	2205
Golden......	Van Winkle.	do.	2	4/27/34	26.7	41.8	51.6	6.6	.4	4.7	68.6	1.1	18.6	8580	11710	12530	2305
Littleton...	Virginia.	do.	2	4/23/34	26.8	40.3	51.0	8.7	1.0	4.4	67.6	1.0	17.3	8430	11520	12620	2200

TABLE 1.- Analyses of Mine Samples, Northern Colorado Coals - Continued

Where mined			Number of samples mixed and analyzed	Date sampled	Moisture, as received	Dry coal								B.t.u., as received	B.t.u., dry coal	B.t.u., ash- and moisture-free	Softening temperature, °F.
County and town	Mine	Bed				Volatile matter	Fixed carbon	Ash	Sulphur	Hydrogen	Carbon	Nitrogen	Oxygen				
Larimer County																	
Carr........	Backy....	Unnamed.	2	4/19/34	32.4	40.0	46.5	13.5	3.0	4.5	63.6	1.4	14.0	7420	10970	12680	2250
Do.	Hackman No. 1.	do.	2	4/20/34	32.6	39.5	50.1	10.4	1.2	4.5	65.5	1.4	17.0	7490	11110	12390	2305
Weld County																	
Dacono......	Baum.....	do.	4	5/21/34	25.6	37.7	57.4	4.9	.4	4.8	73.0	1.5	15.4	9280	12480	13110	2260
Erie........	Boulder Valley (State)	do.	5	5/2/34	22.4	38.6	56.7	4.7	.3	4.9	73.0	1.7	15.4	9780	12610	13230	2090
LaSalle.....	Buddy....	do.	2	4/17/34	30.6	38.0	53.1	8.9	.5	4.8	69.3	1.6	14.9	8260	11900	13070	2230
Erie........	Clayton..	do.	4	5/8/34	22.4	37.8	57.3	4.9	.4	4.9	73.2	1.6	15.0	9810	12640	13290	2120
Serene......	Columbine	do.	4	10/26/32	22.1	39.9	54.5	5.6	.4	4.9	72.3	1.6	15.2	9800	12530	13330	2035
Frederick...	Grant....	do.	4	11/28/32	25.8	37.7	56.4	5.9	.6	4.7	72.4	1.6	14.8	9250	12470	13250	2230
Erie........	Imperial.	do.	4	5/16/34	23.4	37.3	57.0	5.7	.6	4.9	72.8	1.6	14.4	9660	12620	13380	2140
Lafayette...	Monroe...	do.	2	4/12/34	23.1	38.3	56.7	5.0	.4	4.9	73.0	1.6	15.1	9740	12660	13330	2185
Erie........	Morrison.	do.	4	5/15/34	22.2	38.6	54.6	6.8	.4	4.9	71.0	1.6	15.3	9540	12260	13160	2095
Dacono......	Puritan..	do.	4	6/11/34	23.7	37.9	56.5	5.6	.5	4.7	72.1	1.6	15.5	9580	12560	13310	2060
Frederick...	Russell..	do.	3	5/23/34	25.7	38.1	55.5	6.4	.7	4.7	71.9	1.6	14.7	9180	12350	13190	2330
Do.	Shamrock.	do.	2	4/30/34	24.7	37.6	56.3	6.1	.7	4.8	71.8	1.6	15.0	9390	12470	13280	2085
Lafayette...	Standard.	do	2	6/6/34	20.8	37.7	55.6	6.7	.5	4.9	70.6	1.2	16.1	9820	12410	13290	2055
Dacono......	Sterling.	do.	4	6/6/34	25.2	37.9	56.0	6.1	.5	4.7	71.5	1.7	15.5	9310	12450	13250	2100
LaSalle.....	Sunset...	do.	2	4/18/34	29.4	37.4	54.0	8.6	.6	4.7	69.1	1.6	15.4	8350	11840	12950	2090
Ft. Lupton..	Witherbee (Silver State)	do.	2	6/8/34	25.8	38.5	53.5	8.0	.5	4.7	68.4	1.5	16.9	8910	12010	13050	2105

TABLE 2.- Analyses of Delivered Coal; Samples Collected at Mine Tipples, Northern Colorado Coals

Where mined: County and town	Where mined: Mine	Where mined: Bed	Size of coal	Date sampled	Moisture, as received	Dry coal: Volatile matter	Dry coal: Fixed carbon	Dry coal: Ash	Dry coal: Sulphur	B.t.u., as received	B.t.u., dry coal	B.t.u., ash- and moisture-free	Softening temperature, °F.
Boulder County													
Erie........	Arrow (Star).	Unnamed.	2-inch slack..........	4/6/34	23.7	34.4	53.1	12.5	1.1	8840	11590	13250	2230
Lafayette...	Black diamond.	do.	6- to 2½-inch nut coal.	3/27/34	19.9	38.4	56.0	5.6	.5	9990	12470	13220	2110
Gorham......	Crown....	do.	2½-inch slack..........	4/3/34	18.4	37.8	55.9	6.3	.4	9980	12230	13060	2060
Do.	Eldorado.	do.	6-inch lump............	3/14/34	17.0	38.5	55.4	6.1	1.7	10220	12300	13100	2020
Superior....	Gorham...	Gorham..	do.	3/9/34	18.9	38.4	56.1	5.5	.4	10130	12490	13210	2130
Do.	Industrial.	Unnamed.	do.	11/17/32	18.5	41.3	53.2	5.5	.3	10170	12480	13200	2100
Do.	do.	do.	6- to 2½-inch nut coal.	11/14/32	17.7	39.2	55.0	5.8	.2	10200	12390	13150	2090
Do.	do.	do.	Under 2½-inch slack....	11/12/32	17.0	36.3	54.5	9.2	.4	10050	12110	13340	2190
Gorham......	Lewis No. 2.	do.	6-inch lump............	3/19/34	16.3	39.5	55.6	4.9	1.1	10550	12600	13250	2360
Broomfield..	Monarch No. 2.	do.	2½-inch slack (crushed)	6/19/34	16.0	37.8	55.5	6.7	.4	10250	12200	13080	2110
Superior....	Pluto....	do.	6-inch lump............	3/9/34	18.9	37.2	57.1	5.7	.3	10090	12440	13200	2090
Gorham......	Rosser...	do.	do.	3/16/34	16.5	38.5	56.6	4.9	1.0	10500	12570	13220	2280
El Paso County													
Colorado Springs.	City.....	Fox Hill	Pea slack under 1 inch.	7/7/34	21.4	40.7	45.4	13.9	.4	8390	10670	12400	2180
Pikeview....	Pikeview.	do.	Crushed run-of-mine....	7/2/34	22.0	41.6	51.3	7.1	.4	8890	11400	12270	2270

TABLE 2.- Analyses of Delivered Coal; Samples Collected at Mine Tipples, Northern Colorado Coals - Continued

Where mined: County and town	Where mined: Mine	Where mined: Bed	Size of coal	Date sampled	Moisture, as received	Dry coal: Volatile matter	Dry coal: Fixed carbon	Dry coal: Ash	Dry coal: Sulphur	B.t.u., as received	B.t.u., dry coal	B.t.u., ash- and moisture-free	Softening temperature, °F.
Jefferson County													
Leyden......	Leyden No. 3.	Unnamed.	Egg under 6-, over 2½-inch.	5/11/34	20.0	38.3	55.2	6.5	0.6	9540	11920	12750	2140
Do.	do.	do.	Under 2½-inch slack......	5/14/34	20.1	39.2	53.8	7.0	.7	9500	11890	12790	2090
Littleton...	Virginia.	do.	-1¼- and +½-inch steam or nut.	4/24/34	23.4	39.6	50.7	9.7	.9	8690	11330	12560	2140
Weld County													
Erie........	Boulder Valley (State).	do.	-2½-inch slack..........	5/4/34	21.9	38.6	56.8	4.6	.4	9890	12660	13280	2030
Do.	Clayton..	do.	Egg (nut) under 6-, over 2½-inch.	5/9/34	20.0	37.8	56.4	5.8	.4	10030	12530	13300	2050
Serene......	Columbine	do.	6-inch lump.............	10/31/32	20.7	37.8	56.4	5.8	.3	9980	12580	13350	2040
Do.	do.	do.	Pea coal, 2½- by 1¼-inch.	11/1/32	19.7	38.5	54.6	6.9	.3	9960	12410	13330	2050
Do.	do.	do.	Slack under 1¼-inch......	10/26/32	20.4	38.1	53.9	8.0	.4	9770	12270	13340	2140
Frederick...	Grant....	do.	4-inch lump.............	12/23/32	24.0	36.8	57.5	5.7	.6	9550	12570	13330	2200
Do.	do.	do.	Slack under 1½-inch......	12/19/32	25.2	38.2	54.7	7.1	.8	9240	12350	13290	2170
Erie........	Imperial.	do.	2½-inch slack crushed....	6/21/34	19.9	37.6	56.7	5.7	.5	9980	12460	13220	2130
Lafayette...	Monroe...	do.	Slack under 2½-inch......	4/13/34	20.2	37.0	54.3	8.7	.5	9680	12130	13290	2160
Erie........	Morrison.	do.	2½-inch slack (crushed)..	5/17/34	20.3	38.4	54.3	7.3	.5	9660	12120	13070	2090
Frederick...	Russell..	do.	2½-inch slack...........	5/24/34	23.7	38.7	53.9	7.4	.7	9380	12290	13270	2130
Do.	Shamrock.	do.	-2½-inch slack..........	5/1/34	20.5	37.0	57.2	5.8	.6	10050	12640	13430	2080
Dacono......	Sterling.	do.	2½-inch slack(crushed)...	6/12/34	22.0	37.3	55.2	7.5	.6	9600	12300	13290	2050

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DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

FALLS OF COAL AND ROCK ON MAN-TRIPS IN BITUMINOUS-COAL MINES

BY

C. W. OWINGS

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INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

FALLS OF COAL AND ROCK ON MAN-TRIPS IN BITUMINOUS-COAL MINES[1]

By C.W. Owings[2]

INTRODUCTION

A few of the more progressive mining companies have taken precautions to safeguard employees on man-trips. Specially constructed man-cars have been provided with special devices or mechanisms to stop the cars on slopes, if the hoisting cable should break. Cars have been equipped with safety chains to connect and hold them together if a coupling should break. Steel frames covered with heavy wire netting have been placed over cars to reduce injuries to the riders from roof falls on slopes. A safety cable passing from the front car to a point above the cable socket on the slope hoisting cable is used frequently as a precaution in case the socket loosens or the main cable breaks at the socket.

Such devices are commendable, but the percentage of man-cars and man-trips protected is relatively small - entirely too small. Man-trips that travel level haulage roads are rarely protected with safety devices, and haulage roads are seldom inspected by mine officials to determine the condition of roof and ribs.

During the past few years several accidents in bituminous-coal mines have resulted in death and serious injury to men riding on man-trips. Officials investigating such accidents have tended to classify them as "acts of God", ignoring the possibilities of human failure. Such accidents are regrettable, and circumstances often indicate carelessness, ignorance, or both. It is believed that each accident can be a lesson in accident prevention; this circular therefore describes a number of accidents and indicates probable causes and ways of preventing similar ones.

DETAILS OF ACCIDENTS

1. Fall of Roof on Man-trip

A fall of roof on a man-trip killed 3 miners and injured 8 others. The coal bed, which averages 90 inches in thickness, is overlain by a gray shale of

[1] The Bureau of Mines will welcome reprinting of this information circular, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6863."

[2] Associate mining engineer, U.S. Bureau of Mines Safety Division, Pittsburgh, Pa.

variable character. In some parts of the mine the shale is massive, comparatively soft, and streaked with cracks and "slips" or so-called "cutters" running in different directions. The accident occurred on the main entry where the track turned into a parallel entry. When the man-trip reached this point it pulled into the crossover and stopped. Three or 4 minutes later a piece of roof approximately 19 feet long, a maximum of 8 feet wide, and 1 foot thick fell from the roof, covering the fifth car from the locomotive and falling partly on the fourth and sixth. A man in the fifth car, noticing a small piece of shale fall from the roof, jumped into the sixth car just before the large piece of roof fell and was uninjured. The rock did not break but "bridged" across the tops of the cars. Three men in the fifth car were killed instantly, each receiving a fracture of the spine. Three other men were seriously injured; one received fractured ribs, another a punctured lung, and the third a fractured pelvis. Five others were injured but less seriously.

Some of the factors contributing to this accident were:

1. The man-trip had been stopping at this point every morning for several days so the men could get out of the trip, yet the mine officials had failed to test the roof or in testing it either had not detected its loose condition or had detected it but had failed to make it safe.

2. The management failed to provide a suitable place for the men to unload from the man-trip.

3. Ten or 12 men were allowed to ride in each car, overcrowding it, which in this instance resulted in the injury of a larger number of men and perhaps added to the severity of the injuries because the men could not move freely.

This accident illustrates a practice common in some coal mines of neglecting to test roof on haulage roads. State laws usually have provisions that require the operating company, through the mine foreman or his assistant, keep travelways, haulage roads, and working places in safe condition. Unfortunately mine officials are usually so anxious to arrive early at the working places that they ride on the locomotives or in empty cars to the sections, where they remain as long as possible and then ride out. If the mine foreman cannot walk along the haulage roads each day he should assign this duty to an assistant; at any rate the roof and sides of haulage roads should be given much more effective supervision than in most mines.

Another lesson to be gained from this accident is that overcrowding man-cars may result disastrously. When men ride on both sides of man-cars they do not have room to avoid falling rock or coal. The fact that the 3 men killed received fractured spines, that 1 man suffered fractured ribs, and that another received a broken pelvis indicates that because the cars were crowded those in the car on which the roof fell were unable to throw themselves on the floor or to jump from the car and hence received serious injuries from direct blows. The rock did not break and fall into the car in this instance, and if the men could have thrown themselves to the bottom of the car they might have been uninjured or only slightly injured.

Overcrowding cars increases the number of men exposed to falls of rock or coal by increasing the number in a restricted area. Five or 6 instead of 10 or 12 men would have been exposed in this instance if the men had been required to ride on one side of the car only; likewise the number and probably the severity of the injuries would have been decreased. The fact that one man was able to escape injury by jumping from one car to another indicates that the others might have been injured less severely if they had had enough room to move freely.

Moreover, men should not be permitted to ride under the trolley wire, as this constitutes a shock hazard.

2. Falls of Rib on Man-trip

Two men were killed and 3 injured by a fall of rib coal and overhanging rock while a man-trip of 5 cars with 2 to 5 men to a car was waiting at a sidetrack for orders to proceed into the mine.

The coal bed averages 3 feet 5 inches in thickness; to obtain adequate height on haulage roads the fireclay bottom had been removed to a depth of 18 to 24 inches. Several years ago the roof had fallen to a height of about 13 feet at the site of the accident. The roof in general is hard gray shale which requires only limited timbering. The coal is overlain by a stratum of cannel coal or so-called "shale" which has a cubical fracture with numerous slips or cracks along the fracture planes. The immediate dark-gray shale roof had cracks corresponding to those in the cannel coal. At the site of the accident the rock above the dark shale dipped at a sharp angle behind the shale, and the contact between the two shales had a "slick" coating.

Fireclay is said to expand when first exposed and then as it absorbs moisture to disintegrate and crumble. It is believed that the fireclay expanded and that the extra pressure exerted on the overlying coal and rock caused cracks. Later the fireclay crumbled and fell, leaving the coal unsupported for about 6 inches at the bottom and for as much as 2 feet near the roof. As the entry at the sidetrack was 18 feet wide and the edge of the fall only 9 feet from a crosscut the roof may have exerted extra pressure, and the constant vibration of passing trips may have loosened the rib gradually. A loaded trip had passed about a minute before the fall occurred.

The fall of coal and rock from the rib gave only slight if any warning. The piece that fell was about 11 feet long, 8 feet high, and 6 to 24 inches thick. The fourth man in the second car received only a scratch on his finger; the fifth man received a fractured skull when a rock about 3 feet long, 12 to 18 inches wide, and 12 inches thick fell from the upper part of the rib and hit his head, knocking it forward against the iron brace on the opposite side of the car. The first man in the third car was hit on the back and sustained a fractured pelvis and foot. The second man was knocked against the opposite side of the car and received a broken pelvis, punctured bladder, and other internal injuries that caused his death. The third man's back was broken. The fourth man was held in his seat by the fallen material which fractured his spine and caused contusions and bruises from which he died. The men in the other cars were uninjured.

The injured were taken from under the rock and coal, given first-aid treatment, and hauled to the surface.

Failure to test the roof and ribs along the haulage road, especially at the sidetrack where it was customary for the man-trip to stop, was largely responsible for this accident.

Roof and ribs on haulage roads and travelways should be inspected and tested daily or at least twice a week. Tests should be made immediately before the beginning of each shift wherever roof has fallen and coal or rock overhangs main entries or travel roads.

Posts with blocks or lagging between post and rib should be set along the rib at man-trip stops to prevent material from falling and injuring the men. Loose roof should be taken down and if doubtful should be supported.

3. Fall of Roof on Moving Man-trip

A fall of roof on a moving man-trip injured more or less severely 10 of 11 men riding in one car. The company has adopted a number of safe practices to safeguard men riding on man-trips; each man-trip is hauled by a locomotive operated by the section foreman. However, the cars were overcrowded with as many as 11 or 12 men in a car; consequently the men were forced to ride on the side under the trolley wire as well as on the side free of the trolley-wire menace.

The coal bed averages $5\frac{1}{2}$ feet in thickness; the draw slate is overlying the bed 6 to 12 inches thick and must be taken down during mining of the coal. A stratum of coal about 8 to 12 inches thick is left to keep air and moisture from the overlying strata of several inches of draw slate, carbonaceous shale, and gray calcareous shale. The management believes that as long as the roof coal remains in place, although sagging or sounding "drummy" when tested, the roof is good and will stay in place. Considerable timbering is required along the haulage road, and the method of setting timbers is good.

The night crew in one section had completed its shift, and about 20 men entered 2 cars, 11 men seating themselves in the first car. The section foreman took his position on the locomotive and started out of the mine. They had gone only a short distance when the foreman saw pieces of roof falling. He opened the controller to the last notch, and the cars jerked forward violently and threw the men sidewise toward the rear of the cars. At about the same time the roof fell, mainly from the right rib; the men riding on that side of the car were less severely injured than those on the opposite side. It was impossible to determine from the investigation whether the roof was all coal at this point and whether the coal and rock in the middle of the entry had fallen earlier or had been in place. Apparently only overhanging material at the roof on the rib was in place and had fallen. The largest piece of rock measured 7 feet 2 inches long, 2 feet 8 inches wide at the widest point, and from about 7 inches in the center to feather edges on all sides. The upper surface was rounded and smooth or "slick."

At the time of the investigation most of the fallen material had been removed, and loose overhead rock had been taken down; the rock on the opposite rib was loose, and the coal roof for about 100 feet outby the scene of the fall was in place but was decidedly "drummy" and vibrated readily when tested with a heavy piece of iron. In several places part of the coal roof was sagging more than an inch in the center of the entry. Previous experience in this mine led the officials to believe that this coal roof was safe. Each day a timberman travels through this entry, and whenever he finds loose or "bad" roof he removes it and sets crossbars; "drummy" and loose coal is often left in place.

This accident occurred during the hot summer months, and the excessive moisture deposited on the roof at the edge of cavities undoubtedly accelerated the deterioration of the calcareous roof rock. Failure to place a crossbar under the edge of coal roof after the roof has fallen probably allows the coal to break loose from the roof rock and moisture to enter, causing the rock to disintegrate, crack, and ultimately break the underlying layer of roof coal.

The belief that as long as coal is in place the roof rock will not fall is undoubtedly erroneous, and this contributed directly to the injury of the 10 men in this instance.

Here, again, overcrowding of the man-trip was responsible for injury to about twice as many men as would have been injured if the men had been required to sit only on the side of the car opposite the trolley wire. By forcing the trolley wire down on the injured men and on the men in the second car the fall easily might have resulted in one or more electrocutions and severe burns.

The lessons to be learned from this occurrence are:

1. All loose and "drummy" roof, especially on haulage roads, should be taken down.

2. When roof is taken down or if roof falls occur two or more crossbars should be placed close together at the edge of the coal roof on each side of the fall.

3. The man who inspects the roof on haulage roads should be instructed to report to the mine foreman all doubtful as well as loose roof.

4. Men should be required to ride on the side of the man-car opposite the trolley wire.

5. All persons, including officials and visitors, should be required to wear hard hats while underground.

SAFEGUARDING MAN-TRIPS

In many mines no precautions are taken to reduce roof falls on haulage roads even after accidents from falling material have resulted in injury to persons on man-trips. The accidents discussed in this circular in virtually every instance were classified as "unavoidable", and few if any precautions were taken to prevent recurrence of similar accidents insofar as could be determined by later investigation. The engineers of the United States Bureau of Mines believe that definite lessons may be learned from each accident and that a large number of accidents from falls of rock and coal on man-trips can be prevented if information is disseminated, especially if the precautions are taken as pointed out to the mining industry.

After a fall of roof coal which killed 1 man and injured 6 others on a man-trip a coal-mine operator in the mid-West required men to go ahead of the man-trip to examine the roof along the roadway. The roof, overhanging rock, and ribs along haulage roads should be tested by the vibration method as well as by the usual sounding method at least once a day before the major shift starts; a competent, reliable employee should be assigned to this job. Any defective or dangerous condition of roof or rib on haulage roads should be remedied promptly.

Roof coal and overhanging rock on ribs at sidetracks, turn-outs, or other places where man-trips stop frequently should be examined and tested before the beginning and end of each shift on which man-trips are run. Failure of roof or coal for a time to fall should not be accepted as evidence of roof stability.

With certain exceptions roof coal should be considered temporary support. It may be left in place in rooms where the coal is extracted in a relatively short time, but on entries it should be taken down unless special roof conditions make removal undesirable. The belief in certain coal-mining districts that coal or other roof is safe even when "drummy" and vibrating under tests has led directly to numerous fatalities and other injuries from falling roof. The rule "if roof is loose take it down, if doubtful or difficult to take down timber it" should not be waived unless special conditions exist.

Wherever the entry roof is taken down the roof at the entrance to rooms should be supported by two or more crossbars placed about 6 to 12 inches apart. If roof is taken down or if a fall occurs on an entry the same method of support should be used on each side of the roof cavity. Roof coal that is not too weak may be held in place for a time by timbering before it has an opportunity to loosen. However, taking down roof coal is the best policy in most mines.

Man-cars should not be overcrowded, and men should be required to sit on the side of the car opposite the trolley and other power wires. This practice reduces the electric-shock hazard, reduces the number of men exposed to falls of rock and coal on man-trips by reducing the number in each car, and it may reduce the severity of injuries or prevent injuries by allowing men more freedom to move and dodge falling material.

The rib and roof at man-trip stops should be made safe by setting posts about 3 feet apart. The posts should have substantial cappieces, and blocks should be placed between posts and rib to prevent rib material from rolling over onto the man-trip.

Men should be required to wear hard hats while underground, including the time while riding on man-trips. The number and severity of injuries received by some of the men in the accidents described might have been greatly reduced if they had worn hard hats.

Investigations of roof-fall accidents indicate that in many mines, especially those east of the Mississippi River, most falls of roof on entries occur during the hot summer months when the intake air has a high moisture content. The relatively cool temperature of the coal and rock surfaces underground causes the moisture to be deposited on the roof and ribs and in the layers of roof exposed in cavities. Several means of preventing deposition of moisture in mine-roof cavities have been developed, but one of the most promising appears to be a method developed in the mid-West. In a section of one intake air course where water was falling in a number of small streams it was observed that the roof was stable, whereas in other air courses a number of falls occurred during the summer months. Assuming that the water "screen" was a definite factor in reducing falls, a water spray was installed at the foot of the intake air shaft. The results were encouraging, and falls of roof were greatly decreased probably because the water spray reduced the dew-point temperature as well as the daily variation in temperature and thereby decreased the action of moisture and temperature on the roof materials. This method could be adopted with profit in many coal mines.

The number of persons injured on man-trips apparently is increasing. The adoption of the precautions suggested in this circular, in addition to other widely used safeguards, should reduce the hazards from material falling on men riding to and from work on man-trips.

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DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

ACCIDENTS IN TENNESSEE COAL MINES

BY

FRANK E. CASH

I.C. 6864
December 1935

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

ACCIDENTS IN TENNESSEE COAL MINES, 1932-34[1]

By Frank E. Cash[2]

The work of the United States Bureau of Mines is concerned primarily with the prevention of accidents in the mining, petroleum, and allied industries. This report analyzes the fatal accidents in the coal mines of Tennessee and gives the pertinent factors in tabular and descriptive form, with interpretations and suggestions to operators, workers, and the State Division of Mines for the future prevention or reduction of accidents in coal mines.

ACKNOWLEDGMENTS

This study was made possible through the cooperation of A. W. Evans, chief mine inspector of Tennessee, and his secretary, Miss Hiliary Penny, who made available the department records and assisted in the segregation and assembly of the data.

Figures covering coal-mine accidents before 1932 were taken from United States Bureau of Mines bulletins; those for 1932, 1933, and 1934 were furnished by the Division of Mines, Tennessee Department of Labor.

PRODUCTION, EMPLOYMENT, AND ACCIDENTS IN TENNESSEE COAL MINES, 1915-34

Table 1 gives production, employment, and accident figures by years from 1925-34, inclusive, and the average for the 10-year period 1915-24. A comparison of the two 10-year periods shows that the fatal-accident experience based on tonnage and exposure was worse during 1925-34 than during 1915-24. Even if the explosion in the Rockwood mine, which resulted in the death of 27 men, had not occurred the experience during the past 10 years would not have been quite as good as in the preceding 10 years.

ACCIDENTS

This report discusses in detail the accidents that occurred in Tennessee coal mines during the 3 years 1932-34, inclusive.

In 1932 the coal mines of Tennessee, as well as those of the United States as a whole, had their best accident experience on record. The State Division of Mines has a record of all compensable accidents, that is, fatalities and total permanent, partial permanent, and temporary injuries causing the loss of more than 7 days.

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2 District engineer, U.S. Bureau of Mines Safety Station, Birmingham, Ala.

TABLE 1. - Production, number of employees, and accidents in Tennessee coal mines, 1925-34

Year	Production	Number of mines	Days active	Employees	Fatal accidents	Compensable nonfatal accidents	Fatality rate per million tons	Fatality rate per million man-hours	Tons per fatality
1925	5,454,011	132	211	8,314	26	174	4.77	1.82	209,770
1926	5,788,741	133	234	7,948	50	232	8.64	3.33	115,775
1927	5,783,367	111	235	7,691	17	194	2.94	1.15	340,139
1928	5,610,959	116	226	7,849	18	155	3.21	1.24	311,719
1929	5,405,464	81	228	7,619	20	121	3.70	1.40	270,273
1930	5,130,428	79	196	7,535	22	131	4.29	1.82	233,201
1931	4,721,548	83	179	7,448	17	265	3.60	1.56	277,738
1932	3,537,882	75	148	6,827	9	134	2.54	.97	393,098
1933	3,709,405	119	130	6,742	14	252	3.78	2.28	264,958
1934	4,104,910	134	137	7,019	21	323	5.11	3.13	195,472
Ave. 1925 -34	4,924,671	---	192	7,499	21.4	198	4.26	1.87	231,554
Ave. 1915 -24	5,670,635	---	206	10,455	21.0		3.70	1.21	270,013

TABLE 2. - Fatal and compensable nonfatal accidents by causes, 1932-34

Cause	1932		1933		1934		Total	
	Fatal	Nonfatal	Fatal	Nonfatal	Fatal	Nonfatal	Fatal	Nonfatal
Falls of rock and coal	4	41	9	70	14	68	27	179
Haulage	4	35	3	83	2	104	9	222
Electricity	1	7	1	3	1	4	3	14
Falls of persons	0	6	0	7	2	25	2	38
Machinery	0	3	0	15	1	16	1	34
Explosives	0	0	0	6	1	2	1	8
Asphyxiation by mine fire	0	0	1	0	0	0	1	0
Handling material	0	19	0	26	0	50	0	95
Flying objects	0	4	0	22	0	16	0	42
Pushing cars	0	9	0	11	0	8	0	28
Tools	0	7	0	8	0	26	0	41
Gas ignitions	0	2	0	1	0	3	0	6
Total	9	133	14	252	21	322	44	707

CAUSES OF ACCIDENTS

Table 2 tabulates by years and causes the fatal and compensable nonfatal accidents for the 3-year period 1932-34, inclusive.

Falls of coal and rock were responsible for the greatest number of fatal accidents, with haulage second, electricity third, and falls of persons fourth; mine fires caused 1 death in 1932 and machinery and explosives 1 each in 1934.

The order is not the same for compensable nonfatal accidents; haulage leads, followed by falls of rock and coal, handling material, flying objects, tools, falls of persons, machinery, pushing cars, electricity, explosives, and gas ignitions in the order mentioned.

Grouping of fatal accidents for the 3 years according to mines reveals that 1 mine had 4 fatalities, 1 had 3, 9 had 2, and 19 had 1 each; the remaining mines, ranging from 45 to 104 according to number in operation, had no fatalities. The mines reporting 1 or more fatalities produced only 2,922,806 tons (25.75 percent) during the 3 years of a total of 11,352,197 tons of coal produced.

CAUSES OF FATALITIES

Although data on fatal accidents are not complete each coal-mine fatality will be described as closely as the data will permit and possible means of prevention will be suggested.

The statistical data on accidents are taken from the Tennessee Division of Mines reports, and the details are quoted from reports of the State inspector (where available) and the superintendent or manager of the Workmen's Compensation Division.

Falls of Rock and Coal

1. A white miner 43 years old, with 25 years' mining experience, was killed by a fall of slate. He was married but had no children.

> The place had just been cut, and in loading the machine the safety post was knocked out and not reset by the machine crew. The deceased entered the place and in all probability did not examine the roof. A piece of slate 9 feet by 6 feet by 2½ feet thick at the face, and tapering to a featheredge back in the entry, broke at the face, killing the miner.

The miner should have examined and made the roof safe before he went under it, the machine crew should have reset the post knocked out, and the management should have required both precautions from its employees.

2. A white miner 49 years old, with 1½ years' mining experience, was killed by a fall of slate. He was married and had 4 children.

> The deceased was loading coal in a room when 12 to 15 tons of rock (horseback) fell, killing him. This seems to be another case where a safety post or posts might have saved a life. Prior to the accident no systematic timbering rules were extablished or enforced.

Safety rules should be established covering testing and taking down or timbering roof, especially in a mine where rolls or "horsebacks" are encountered; the mine foreman should see that the rules are enforced, and the miner should test and make the roof safe.

3. A negro miner 30 years old, 1 year's mining experience, was killed by a fall of slate in an entry. He was married but had no children.

> The entry was 9 feet wide, and a piece of slate 6 feet by 3 feet by 15 inches thick, with slips near each rib, fell on the deceased's head and shoulders, breaking his neck and jaw. There was one safety post under the rock when it gave way and rode the timber.

Responsibility rests jointly on the supervisor and the miner; the rock should have been taken down or timbered more securely.

4. A white miner 22 years old, single, with 5 years' mining experience, was killed by a fall of rock.

> The deceased was working with his father pulling room pillars. The roof had been "working" the previous day, and the foreman instructed the 2 men to move toward the entry about 15 feet, shoot the pillar, and work toward the working top. This was on August 10, and on August 11, due to boiler-water shortage, the foreman remained on the surface and the miners went to their working places. The father started drilling at the point designated by the foreman, and the son, or deceased, came in the mine later and started digging coal inby the father, and a piece of slate 10 feet by 2½ feet by 6 inches thick fell on his head and shoulders, breaking his neck and causing internal injuries which resulted in his death before reaching the hospital. The accident was avoidable, as the draw slate was exposed along the rib, and should have been taken down or timbered with 2 or 3 posts. If the slate had been tested, it would have sounded "drummy." Closer supervision by bosses and closer attention to the roof condition are recommended.

The men should not have been allowed to enter the mine without someone in a supervisory capacity in the mine or with the men. This place should have been examined by the foreman before the men went to work, as he knew the place was unsafe the previous day.

5. A white miner 60 years old was killed by falling slate.

In a recently reopened mine which had not been operated for 23 years, the deceased, assisted by his son, was drawing entry stumps. The son was turning the coal and the father was loading it into the car when the roof fell the width of the entry, 5 feet wide and 2 feet thick, killing the father. Although the rock was too heavy to have sounded "drummy" the accident was caused by removing the coal and not setting timbers to support the roof. Timbers should have been set as the coal was removed.

The miner was a "contractor", that is, mined coal on a royalty basis, and was responsible for his own safety; but, like many other experienced miners when not forced by some supervisory authority, he failed to take adequate and readily available precautions.

6. A white miner 25 years old, with 10 years' mining experience, was killed by a fall of roof. He was unmarried.

The deceased was instantly killed while undermining coal in a crosscut being driven from his room. The room was reasonably well timbered, but timbers were set in a semicircle around the breakthrough. A slip or horseback 8 feet by 4 feet by 10 inches thick fell on the miner. If the roof had been tested, it would have been found loose, and two timbers would have supported the rock. Standard timbering, closer examination by miner, and closer examination and supervision by foremen are recommended.

This description places the responsibility jointly on the miner and foreman; unfortunately joint responsibility is far too common, and as a result many persons are crippled or killed.

7. A miner was killed by a fall of slate in an entry.

The State law requires that the operator notify the State Division of Mines when a fatal accident occurs and that the mining division investigate all fatal accidents. No details of this accident were furnished by the mining company, and no investigation was made by the State Division of Mines.

8. A white miner 57 years old, with 25 years' mining experience, was killed by a fall of slate at the robbing face in an entry. He was married.

Robbing, had timbers close to face; 3-foot by 3-foot by 6-inch piece of apparently sound slate broke, fell around timbers, and killed him.

There are not enough data on this accident to permit definite comment; one would infer that the person making the report considered the accident largely unavoidable.

9. A white machine helper 34 years old, with 5 years' experience in the mines, was killed by a fall of rock. He was married and had 4 children.

> The machine was cutting across the face of a room when the roof fell, killing the machine helper.

A machine crew should not cut a place if the roof is not safe and cannot be made safe. In all probability timbers set behind the machine as it cut across the face would have prevented this accident. Machinemen particularly are inclined to take undue chances with roof conditions, and in consequence their occupation is considered extra-hazardous though usually there is no good reason why these workers cannot protect themselves adequately by expending a little time and effort other than in making cutting records.

10. A white miner 44 years old, with 1 month's experience in mining, was killed by a slate fall. He was married.

> The deceased was working alone and had been instructed to set a timber under a bad piece of slate. It looked as though he had just entered his place when the slate fell on him.

Closer supervision by the foreman or compliance with instructions by the miner in all probability would have prevented this accident. The foreman should not have left a miner with 1 month's experience alone to set timbers under loose top.

11. A white driver 45 years old, with 25 years' mining experience, was killed by a slate fall. He was married and had 5 children.

> Unforeseen accident. Deceased entered room for loaded car, leaving mule on the entry, and while driver was passing the car, slate fell on him and car, killing him.

Posts with crossbars would have prevented this accident, that is, there should have been systematic timbering and closer supervision. This type of accident occurs far too frequently and is the direct result of inadequate supervision of safety conditions on haulage roads.

12. A miner was killed by a fall of slate.

> Two men in mine, which was not worked after the accident.

The data are too meager to permit adequate comment as to the probable underlying cause.

13. A white miner 62 years old, with 20 years' mining experience, was killed by a fall of coal from the rib. He was married and had 1 child.

> The accident was due to the deceased's own carelessness. He had fired a shot and went back to load the coal without noticing the overhanging top coal which fell on him, causing his death.

Although the information indicates that the miner was responsible for his own death, he might have been more cautious with proper training and supervision; moreover the surrounding conditions may have been such that only part of the blame rests on his shoulders.

14. A white miner 45 years old, with 3 months' mining experience, was killed by a fall of slate. He was married and had 7 children.

> There was no safety post set, and a horseback fell without warning, killing the miner.

This accident resulted from insufficient timbering and inadequate supervision; here, again, responsibility is divided. Unquestionably many mines should be supervised far more effectively than they actually are.

15. A white hostler 22 years old was killed by a fall of slate. He was unmarried.

> No other information is available in connection with this fatality.

16. A white machine runner 22 years old, with 5 years' mining experience, was killed by a fall of slate. He was married and had 1 child.

> Slate 19 feet by 8 feet by 8 inches, with 3 sides loose, fell, killing the machine runner. There was supposed to have been one post under the rock, but this is doubtful. This fall also injured the machine helper.

Again, the machine crew did not protect themselves while cutting a place, and the men probably were more intent on cutting the place quickly than on taking precautionary measures to protect their own lives.

17. A white motorman 22 years old, with 1 year's experience in mining, was killed by falling slate.

> The deceased was removing crossbars, and two bars broke and roof fell, killing him.

Removing timbers is a dangerous practice, but the persons doing such work usually can protect themselves if they really try. Apparently a man with only 1 year's experience was assigned to far too dangerous a task, because removing timbers of any kind in a coal mine is one of the most hazardous kinds of work in coal mining and should be entrusted only to careful experienced miners.

18. A white miner 26 years old, with 2 years' mining experience, was killed by falling slate. He was married and had 3 children.

> A "kettle bottom" with no timber under it fell, crushing the miner. This is one of those accidents that just happen. The safety post was within 1 foot of the edge of the rock that killed him.

Although kettle bottoms present very difficult problems in safeguarding coal miners, yet a definite timbering system affords fairly adequate protection.

19. A white miner 25 years old, with 1 month's mining experience, was killed while pulling draw slate. He was married.

> Deceased was working in room with bad top and no safety post. This rock also injured another miner.

Here, again, the information is too meager to permit accurate comment, but unquestionably negligence and inadequate supervision contributed.

20. A miner 43 years old was killed by f lling slate.

> An unavoidable accident. The deceased was robbing in coal 9 feet thick, the place was properly timbered, but due to weight, a rib roll of coal and slate caught and killed the miner. This is apparently one of those accidents which just happen.

If timbers will not give the proper protection to workmen while withdrawing pillars a coal pillar should be left for protection, and if the region is so badly broken that adequate safety measures cannot be taken the region should not be worked.

21. A white miner 35 years old, with 10 months' mining experience, was killed by falling slate. He was married and had 5 children.

> The accident occurred at 12:15 p.m. The place had been visited by the foreman at 8:15 a.m. and deceased instructed to set a safety timber. The timber was not set, and the rock which killed him was 6 feet by 1 foot by 18 inches thick.

In this instance supervision failed, and the victim paid in full for his negligence and for defiance of instructions and breach of discipline.

22. A white miner 20 years old, with 6 months' experience in mining, was killed by falling slate. He had been married only 1 week.

> The accident is a result of carelessness and violation of the rules.

> Slate 7 feet 10 inches by 6 feet 6 inches by 10 inches thick, with slips on 3 sides and held at 1 corner, fell, killing the miner. I am reasonably sure there were no timbers under the rock, and also certain that 1 timber would have held it. The foremen are not enforcing the timbering rules.

This accident resulted from failure of the miner to protect himself and from inadequate supervision.

23. A white miner 62 years old was killed in a room break-through by falling slate. He was married and had 4 children.

Working in a break-through between No. 1 room and first left air course, the deceased was lying down working out the last cut in the break-through when slate 12 feet by 10 feet by 2 feet thick fell on him. The room was well-timbered, but in all probability there were no timbers in the break-through.

The miner failed to protect himself, and the foreman allowed a break-through to be driven without being properly timbered - another instance of negligence by the victim and of defective supervision.

24. A miner was killed by falling slate.

Horseback 17 feet by 9 feet fell, crushing deceased. The room was well-timbered outby face.

The responsibility lies with the miner for not properly timbering and with the supervisor for not having the necessary timbers set - a repetition of numerous other instances of negligence by the victim and of inefficient supervision by the operating offender.

25. A white miner 40 years old was killed by falling slate. Additional data are not available on this accident.

26. A white miner 40 years old was killed by falling slate. He was married and had 4 children.

This mine was closed in February on account of not having a certified foreman. In March the son of the original operator was robbing pillars when slate fell on him, causing his death 4 days later.

Evidently the accident was due to negligence.

27. A white miner 52 years old was killed by falling slate. Other information is not available on this accident.

Haulage

1. A white miner 43 years old, with 20 years' mining experience, was run over and killed when a trip of loads broke loose. He was married and had 3 children. Additional information is not available on this accident.

2. A white miner, 54 years old was run over and killed by an electric locomotive. He was married and had 1 child.

The deceased was walking into work on the main entry when he was overtaken, run over, and killed by a motor.

Man-trips were not operated, and a manway was not provided; therefore the men walked along the main haulage. The accident was due to lack of control of locomotives or to improper supervision; the victim also may have been negligent. A man-trip or a manway would have prevented this accident.

3. A white pumpman 26 years old, with 3 years' mining experience, was run over and killed by an electric locomotive. He was married and had 2 children.

> Deceased was riding on front end of motor when his foot in some way slipped, causing him to fall in front of the motor. The motorman could not stop and ran motor up on him, fracturing skull, arm, and leg, from which he died 2 days later. At the point of the accident on the entry the entry was 4½ feet high and 13 feet wide. There was a 2-foot clearance on the side opposite the trolley wire. The wire was 48 inches above the rail.

Although many (probably most) mining companies allow certain employees in addition to haulagemen to ride on locomotives in mines, the practice is by no means a safe one.

4. A white coupler 27 years old, with 6 or 7 years' mining experience, was killed while rerailing a wrecked mine car. He was married and had 2 children.

> Mine car wrecked, coupler placed a 4-inch by 6-inch tie to rerail the car, and when the motor pulled, the car slipped off the tie, catching the coupler between car and rib, causing internal injuries from which he died 6 days later. This was carelessness for an inexperienced coupler.

Rerailers should be provided for putting wrecked cars back on the track, and the person placing them should get out of the way before the cars are moved.

5. A white coupler 39 years old, with 1 year's mining experience, was killed when caught between loaded car and roof. He was married and had 5 children.

> The grade made on the haulage made it necessary to double the trip. The deceased jumped between the cars while the trip was in motion to cut the rear half of the trip off. In getting on the trip it is assumed that his head hit the roof, and he was rolled between top of loaded car and roof. The entry is 46 inches above the rail and 12 feet wide. It is recommended that trips be stopped before they are uncoupled.

The inspector's recommendation to stop trips before attempting to uncouple them no doubt would have prevented this accident; haulagemen are prone to take chances; they should be given careful instructions as to safe procedure in their work, and the instructions should be rigidly enforced.

6. A night foreman 33 years old, with 2 weeks' foremanship experience, was caught between car and rib, causing internal injuries which resulted in his death. He was married and had 3 children.

> Night foreman, acting as coupler, jumped from motor to close door and was rolled between car and rib. The clearance was not sufficient to clear man between car and rib. The door should have been closed by turning latch from motor.

A ventilating door should never be fastened, but held open. Adequate clearance should be provided where men get off trips to open doors; and certainly no underground employee should be allowed to jump off moving haulage equipment in mines except in an emergency to save his life. Jumping on and off moving cars and locomotives in mines is common practice but is extra-hazardous and is absolutely unnecessary in the normal working of mines.

7. A white coupler was injured by wrench between locomotive and roof, causing his death.

> The deceased made up a trip of 17 cars and got on the front end of the motor. The trip had just started when the motorman heard the coupler say "Stop." The trip was stopped, and it was found that a bell wrench in the coupler's pocket had ruptured an artery in his abdomen, causing his death. It is presumed the wrench forced him against the roof, which was 20 inches above bumper and 5 inches above top of motor.

The front end of a mine locomotive is a very unsafe place on which to ride.

8 and 9. Two miners, one 17 and the other 23 years old, were killed when a man-car rope broke.

> These men were told by the hoistman not to get on the car because the hoist drum was not safe. Four men got on the car as it was being lowered, the drum broke, breaking the $\frac{1}{2}$-inch hoist rope, and the car ran down the surface incline, killing 2 men and injuring 2 others.

If the hoist drum was known to be unsafe the car should not have been moved; this apparently is a clear case of disobedience by workers and of lack of discipline and of supervision by the operator.

Electricity

1. A white laborer 22 years old, with 2 years' mining experience, was electrocuted. He was married and had 1 child.

> The motorman and laborer, working as a coupler, were loading and hauling slate from the mine to the surface. It was necessary

for the locomotive to stop while the foreman and trackman put in a rail. The motorman and deceased decided to eat lunch, the motorman against the rib and the laborer on the locomotive. While eating, the deceased came in contact with the trolley wire. The motorman, foreman, and trackman gave artificial respiration, but the victim probably died from strangulation. The wire is 51 inches above the rail, and the locomotive 33 inches high.

A man should not use a locomotive for such purposes, especially under such low top. The foreman who was only 50 feet away should not have allowed the laborer to sit on the locomotive to eat lunch. The wire was less than 6 feet above the rail and should have been shielded. Before artificial respiration was attempted all foreign objects should have been removed from the mouth.

2. A white surface or tipple motorman 28 years old, with 4 years' mining experience, was killed by coming in contact with a 250-volt trolley wire. He was married and had 2 children.

Deceased was standing, one foot in the deck and the other on the bumper of the locomotive, with his back in the direction of motion. He was backpoling, and his clothing was wet. His shoulder contacted the trolley wire. Artificial respiration was given for 2 hours without success.

Standing in and on a locomotive with the back in the direction of motion is a very unsafe combination for locomotive operation; backpoling is also unsafe and usually unnecessary.

3. A white trackman 22 years old, with 5 years' mining experience, was electrocuted by coming in contact with a 250-volt trolley wire. He was married and had 3 children.

The deceased contacted the trolley wire with his neck as he was stepping into the deck of the locomotive. Artificial respiration was given 1 hour and 15 minutes without success.

A low, unshielded trolley wire at the intersection of two entries caused this accident; these trolley-wire accidents indicate the desirability of having the trolley wire shielded at all points where less than 6½ or 7 feet above the track rail.

Explosives

1. A white miner 22 years old with 5 days' mining experience, was killed by using short fuse with explosives. He was married and had 2 children.

The deceased placed a 2-foot fuse in a stick of pellet powder, lit it, and then with a tamping bar pushed it back in a 6-foot shot hole. Evidently the bar forced the lighted end of the fuse against

the charge, as the explosion occurred before the bar was removed. The flying coal killed one man and injured another. Suicide.

Ignorance and lack of training and supervision contributed to this accident; moreover, the use of pellet powder is unsafe in any mine, and short fusing is anything but safe and apparently must have been more or less the usual practice in this mine.

Mine Fire

1. A white slate loader 25 years old, with 1 month's mining experience, was overcome by smoke and gas from a mine fire and fell on the haulageway, and a locomotive ran over him.

The deceased was loading slate near a mine fire. After being overcome by smoke and gas, and probably dead, a locomotive ran over him.

It certainly is a reflection on the supervisory force to place a man with 1 month's mining experience where he could breathe the return air from a mine fire.

Machinery

1. A white ash roller 24 years old, who had been at work for 3 months, was scalded to death by steam and hot water. He was married and had 1 child.

The deceased was working in the ash pit of a steam boiler plant when the "L" on the blow-off valve to the boiler broke, and the steam and hot water ran into the ash pit, causing his death.

This accident might have been prevented by proper boiler inspection.

Falls of Persons

1. One white and one colored miner and four other workmen were killed when a scaffold on which they were working in the construction of a tipple collapsed.

There were 24 men constructing a tipple. At the end of the previous day the construction engineer inspected the structure and was satisfied with its condition and progress. It was inspected an hour before the accident and pronounced safe. While placing the permanent braces, No. 6 bent began to fall, due to loosened guy wires, and knocked the other five bents, with the scaffolds, down.

The guy ropes are said to have been loosened intentionally; if this is a fact this accident should be termed "multiple murder."

NONFATAL COMPENSABLE ACCIDENTS

Haulage

During the 3-year period 22 compensable nonfatal accidents causing 8 or more days lost time and a number of partial permanent disabilities were charged against haulage.

These accidents were due to various causes, such as coupling cars while in motion, being caught by wrecked cars or trips, rerailing wrecked cars, blocking or scotching cars, and being caught between car and timbers or ribs; other causes accounted for a few accidents.

The changes or improvements that would reduce materially the frequency of haulage accidents are:

1. Repairand aline mine tracks
2. Repair mine cars and locomotives.
3. Provide rerailers and require their use.
4. Provide necessary clearance.
5. Provide adequate supervision.
6. Train all employees in accident prevention.

Pushing Mine Cars

Pushing mine cars caused only 28 accidents during the 3 years, but this practice is so closely related to the occurrence of accidents of considerable severity, such as strained back, hernia, and rupture, that it deserves serious consideration.

In some Tennessee mines, empty and loaded cars must be pushed excessive distances, over undulating floors; such practice results in far too many accidents. The remedy is obviou.

Handling Material

Ninety-four accidents were attributed to handling material, largely dropping coal or slate on toes and catching hands between coal or slate and cars. These accidents could be reduced at least 50 percent by the use of safety shoes.

Flying Objects

Ninety percent of the 42 accidents from flying objects were due to coal or slate in eyes. These could be virtually eliminated by the use of goggles; some mining companies now require the use of goggles with lenses to correct defects of vision and thus eliminate not only eye but also numerous other types of accidents through improved vision of the workers.

Other Causes

Other causes of accidents were: Tools, 41; falls of persons, 38; machinery, 34; electricity, 14; explosives, 8; gas ignitions, 6.

Accidents from all these causes could be reduced by closer supervision and by properly selecting, placing, and training employees.

SUMMARY

The fatal as well as the nonfatal accident rate for Tennessee coal mines is better than the average rates for the United States. This record is no reason, however, for not improving the accident experience or for allowing it to be poorer in the past 10 years than in the previous decade. Compared with other mining States, natural conditions in Tennessee favor safe operation: Few mines liberate explosive gases; virtually all coal mined is above water level, and little pumping is required; the coal cover on the average is not excessive, enabling a high percentage of recovery at relatively low cost for roof support; and, with exceptions, roof conditions are fair to good.

CONCLUSIONS

To improve accident experience or reduce accidents in Tennessee coal mines operators, miners, and the State Division of Mines must cooperate.

Operators should put their mines on the most efficient operating basis possible, that is, a safe operating basis. They should formulate a set of safety and operating rules and provide the supervisory force necessary to enforce them. The operators should provide the necessary training; employees thus trained will usually be careful and efficient.

Employees, workmen, or miners should cooperate with employers in reducing accidents to the extent of conforming to the safety and operating rules and availing themselves of all possible training, such as first aid, mine rescue, and accident prevention, and attendance at regular safety meetings, which will enable them to become safer and more efficient workers.

The State Division of Mines of the Department of Labor should employ an adequate number of district inspectors to make thorough and complete examinations of all mines in the State at least every 60 days and forward reports of such inspections to the chief of the Division of Mines, who will forward copies with his recommendations to operating heads of companies and to superintendents or mine foremen. These recommendations should be later followed by inspection to see that they are carried out.

The State Inspection Division should investigate each fatal and serious accident as soon as practicable after its occurrence and send a report placing the responsibility and recommending ways of preventing similar accidents to the operating head of the company and superintendent or foreman of the mine concerned.

After this report has served your purpose and if you have no further need for it, please return it to the Bureau of Mines. The use of this mailing label to do so will be official business and no postage stamps will be required.

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RECENT TRENDS IN DESIGN AND CONSTRUCTION OF COPPER CONCENTRATORS IN THE SOUTHWEST

BY

C. E. Rork

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

RECENT TRENDS IN DESIGN AND CONSTRUCTION OF COPPER CONCENTRATORS IN THE SOUTHWEST[1]

By C. E. Rork[2]

INTRODUCTION

This paper discusses various mechanical improvements that have been gradually incorporated in the copper-milling plants of the Southwest during the past 14 years, power plants for concentrators, use of large-size units of equipment, and the effect of plant capacities on costs.

The improvements in mechanical equipment and plant design have contributed in the reduction of operating costs; in some instances a saving in the capital cost per ton of daily capacity milled has resulted.

TRANSPORTATION EQUIPMENT

Pan and Apron-Type Ore Feeders

Pan and apron-type ore feeders usually are made of steel-plate pans or plates articulated with each other and mounted on two parallel endless chains provided with rollers at the pitch points and supported by rails. Under the strain of continuous heavy service the replacements of worn wheels and bent pans for feeders of this type have proved costly.

Modern heavy-duty feeders are equipped with heavy, cast manganese-steel plates secured to two endless manganese-steel chains. The chains ride on rollers mounted in pairs on shafts. The plate and chain articulation is so arranged that the chain-joint flexure comes to a stop when the plates are level with each other on the load-carrying side. This action enables the assembly to support the load between rollers. Roller wear has been minimized because the design allows the use of shafts and bearings of ample size and provides for proper lubrication. The manganese-steel plates are rigid, and replacements seldom are required for 2 or 3 years. A saving thus is effected in maintenance costs, and shut-downs due to feeder troubles are virtually eliminated. A feeder of this type is used at the New Cornelia crushing plant of the Phelps Dodge Corporation and at the Sacramento shaft of the Bisbee branch of the same corporation.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgement is used: "Reprinted from U.S. Bureau of Mines Information Circular 6866."

2 One of the consulting engineers, U.S. Bureau of Mines.

Modern pan and apron-type ore feeders of manganese-steel construction are strong enough to resist the impact of falling ore. Capacities usually can be increased 50 percent by increasing the height of the skirt boards and carrying a deeper bed of ore. Apron speeds of these machines are usually 20 to 30 f.p.m. and can be increased 20 percent to gain additional capacity.

Belt Conveyors

The outstanding improvement in the design of belt conveyors in the mills of the Southwest is the use of antifriction idlers. Very few mills are completely equipped with conveyors of this type; in the majority of cases they have been purchased as replacement equipment in the last 5 or 6 years. Compared with the plain-bearing-type conveyor, antifriction idlers have effected a substantial saving in power consumption, maintenance cost, and labor and supplies incident to lubrication.

The plain-type-idler installations require frequent lubrication, whereas one filling of grease in the antifriction type usually suffices for [illegible] or 6 months.

The power required for level conveyors equipped with anti-friction idlers and bearings is conservatively half that required for those equipped with plain bearings.

Belt conveyors are rated according to the number of cubic feet of ore handled per hour at a belt speed of 100 f.p.m. This amounts to $3.5\underline{A}^2$, where $\underline{A}$ is the width of the belt in inches. In actual design, however, the capacity allowance for conveyors traveling at 100 f.p.m. is $3\underline{A}^2$, providing for an increase of capacity of 16 percent without a change in the speed.

Modern antifriction conveyors can be run at a higher speed than those with plain bearings, when carrying the average sizes of ore. The following maximum speed limitations are advisable.

Size of ore, inches	Maximum speed, f.p.m.
16	200
12	250
6	300
4	400
3/4	600

The tabulation that follows gives the cost per foot of conveyor for belt and idlers only and the percentage increase in capacity over the preceding size. These costs are for average-duty equipment and do not include costs of altering supports or terminal.

Width of belt, inches	Cost per foot			Increase of capacity percent
	Idlers	Belt	Idlers and belt	
18	$ 9.30	$ 5.25	$ 14.55	--
20	9.40	5.80	15.20	23
24	9.75	6.45	16.20	43
30	10.80	8.65	19.45	56
36	13.80	11.00	24.80	44
42	14.50	13.50	28.00	36
48	15.50	17.00	32.50	30
54	16.75	18.75	35.50	26
60	22.75	21.50	44.20	23

Pumps for Mill Pulps

Centrifugal pumps employed in transporting mill pulps are now built with thick impellers and liners, with provisions for taking up the clearance caused by wear. As wear is unavoidable this arrangement allows a pump to remain in service longer than would otherwise be possible. The wearing parts usually are made of fairly hard cast iron to reduce the cost of renewals.

The principal improvement in these pumps is the outgrowth of more efficient metallurgical practice in the foundry. The wearing parts formerly were made of hard, white iron produced from a mixture of pig, scrap cast iron, and scrap steel. Virtually the same ingredients are used now, but the chemical constituents are balanced carefully to produce an iron of known properties. In consequence, the parts of the pump are tough and strong and have superior wearing qualities. This feature has contributed materially toward reducing maintenance costs.

Failure of the stuffing boxes of sand pumps often causes delay. Designs incorporating improved stuffing boxes, ejectors which replace stuffing boxes, and suction intakes without them are now available.

Bucket Elevators

Many bucket elevators have been replaced by sand pumps in the copper mills of the Southwest. Finer grinding, a practice that is growing in favor, has made these practical, and lower maintenance cost and fewer delays have made the change desirable. However, a survey of recent information circulars published by the United States Bureau of Mines shows that some bucket elevators have been retained in the majority of southwestern copper concentrators, as indicated in the following tabulation:

Use of bucket elevators in southwestern concentrators

Concentrators	Class of service
Hayden	In coarse crushing plant, fine crushing plant, roll circuits, and handling concentrates.
Cananea	Sampling plant.
Magma	Handling Marcy mill product, table middlings and tailings, and secondary grinding-mill discharge.
Nacozari	Handling concentrates.
Hurley	Coarse crushing, handling grinding-circuit products and concentrates.
Copper Queen	Handling concentrates.
Old Dominion	In coarse crushing plant, 1 of 2 units.
United Verde	No elevators.
Miami	Do.

Belts usually are 8- to 12-ply, with 1/8- to 3/32-inch rubber cover. Standard practice is to have them wide enough to allow a margin of 1 inch on each side of the buckets.

The buckets, manufacturers' standard style A, are of malleable iron with reinforced lips. They are spaced on 12- to 30-inch centers and mounted in single row, double row, or double row staggered, according to the volume of material to be elevated.

The most important recent improvement in belt-and-bucket-elevator practice is the application of enclosed speed reducers for driving, which eliminate undue wear from grit.

Chutes, Launders, and Flumes

The outstanding recent improvement in chutes for dry ore is the use of rubber lining. The rubber liners are of special composition and are 1/4 to 3/4 inch thick, according to the size of ore handled and height of drop.

The greater the impact the thicker the lining required to provide the necessary resilience. Lower costs per ton of ore handled are claimed for such liners when installed properly under the required conditions than for the various irons and steels previously used; these claims have been substantiated in many instances.

The launders now employed are of the same general construction as those used for many years. They are usually built of 2-inch lumber, dressed to 1 5/6 inches, and are 3 1/2, 5 1/2, 7 1/2, 9 1/2, or 11 1/2 inches wide inside.

In the days of gravity concentration the maintenance of launder liners was a problem of considerable importance. With the introduction of flotation requiring finer grinding the cost of maintaining liners has decreased. A comparatively recent innovation is the use of discarded conveyor and elevator belting for launder liners in places where the wear is noticeable.

The outstanding improvement in the distribution of pulps from the classifier overflows to the flotation machines is the practice of using pipes instead of launders. Flotation middlings also are handled in pipes in many plants. The following copper concentrators of the Southwest, described in information circulars of the Bureau of Mines, use pipes.

Concentrators using pipes for conveying pulps

Concentrator	Service
Copper Queen	Conveying classifier overflow pulps to flotation and rougher concentrate pulps to cleaner cells.
Morenci	Distributing tailing pulps to form dams.
Hurley	Conveying tailing pulps to ponds; 30-inch wooden-stave pipeline 6,590 feet long.
Magma	Conveying concentrate pulp from mill to thickener at smelter; 4-inch-diameter pipeline 2,900 feet long.

TRANSMISSION EQUIPMENT

Belt Drives

Belt drives from line shafts have been giving way to direct motor drives during the last 10 to 12 years; this change has resulted in savings in maintenance costs, economy in power due to higher mechanical efficiency, and reduction in operating hazards.

The short-center belt drive used for the last 8 to 10 years has been popular. In this type of drive the angle of contact of the belt with the smaller pulley is maintained by the use of an idler pulley running on the outside of the belt. This pulley is mounted on a hinged lever arm and is provided with a holder on which to place the proper weight.

Speed-Reducer Units

During the last decade enclosed-gear speed reducers have been used extensively. Previously they were used in only a few mills because they were obtainable only on special order to suit special conditions.

Speed reducers of various sizes and ratings have been developed and are now on the market. They are cataloged in closely consecutive speed ratios ranging from 1:1.5 to 1:1,200 and in horsepower ratings from 1/2 to 250. There are three general types in use: Enclosed worm gearing; enclosed spur, helical, or herringbone gearing; and planetary gear type.

These reducer sets are built in oiltight cases; a charge of oil provides sufficient lubrication for several months. The wearing parts are protected from dirt, and the design conforms to safety regulations.

The mechanical efficiencies are high, approximately 90 to 95 percent on speed ratios of 1 to 20 or 30.

In the copper mills of the Southwest these speed-reducer sets are used for reducing the comparatively high speeds of electric motors and transmitting the power to ball and rod mills, classifiers, ore feeders, conveyors, and elevators.

Gears with Generated Teeth

Up to 20 years ago millmen were satisfied with plain-cast tooth gearing. Since then the general practice has been to use machine-cut gears, the teeth of which are formed by milling cutters. The shape of these cutters depends on the number of teeth to be cut in the gear; compliance with the market demands for gears of 12 to over 200 teeth in each of the various pitches would require the manufacturer to be equipped with 2 or 3 thousand of these tools. To reduce the number of tools required it became standard practice to use each cutter for forming gears with a range of several steps in the number of teeth; for example, a tool made to cut 50 teeth could be used to make gears of 44 to 56 teeth. This obviously resulted in inaccurate tooth profiles.

Since 1920 machines have been designed to generate a true involute tooth curve on gears of any size or pitch and with any desired tooth angle. Plain spur, helical, and herringbone gears are generated by these modern methods. The generated gears run quietly and smoothly and have an efficiency of 97 percent per pair.

Generated spur gears are used on the 8 1/2-foot ball mills in the Copper Queen and Morenci concentrators of the Phelps Dodge Corporation. These gears have a diametrical pitch of 2 15-inch faces, and the ring gears have 240 teeth. Pinions of 13 and 14 teeth, respectively, are used to change the mill speed. The pinion shafts are connected by flexible couplings to 300 hp. synchronous motors having a speed of 225 r.p.m. The average motor input is around 240 hp.

V-Belt Drives

An efficient transmission device recently designed is the V-belt drive; it consists of two grooved pulleys and one or more endless V-belts. The belts are trapezoidal in cross-section, shaped like a truncated V, with the sloping sides at an angle of about 40° from each other. They are made of cotton fabric and cords impregnated with rubber on a rubber core, the whole being vulcanized in a mold. Various lengths of rope are available to suit different pulley centers. The sheave-groove faces have the same angle (about 40°) as the ropes. Clearance is provided at the groove bottom to give a wedging grip.

These drives are available in capacities of 1/2 to 300 hp. and in reduction ratios of 1:1 to 7:1. Five standard cross-sections of rope are used, ranging from 1/2 by 11/32 inch to 1 1/2 by 1 inch. The number of ropes used on a single drive ranges from 1 to 20.

The advantages of V-belt drives may be summarized as follows: (1) Low maintenance cost; (2) Low first cost; (3) low power loss; (4) silent operation; (5) very short centers although practical without idlers; (6) easy replacement

of ropes; (7) can be run in either direction and at any angle; (8) speed ratio can be readily changed by changing one pulley; (9) resilience of ropes avoids starting shocks; (10) ultimate slipping of ropes prevents damage if the machine becomes locked.

The cost of V-belt drives can be estimated approximately from the formula: Cost in dollars = $\frac{7,200\ \underline{R}}{\underline{S}}$ + hp., where R is the drive-speed ratio, S the motor speed in r.p.m., and hp. the horsepower of the motor.

Antifriction Bearings

As with other modern transmission machinery, antifriction bearings are being utilized in the copper concentrators of the Southwest. Three general types of antifriction bearings are in use - balls, conical rollers, and cylindrical rollers. The ball bearings and conical roller bearings absorb both radial and axial thrusts, whereas the cylindrical roller bearings take radial thrust only.

The advantages in the use of these bearings are in the elimination of friction losses and reduction in expense of lubrication.

In addition to the use of these bearings on conveyor equipment, they have been found extremely advantageous as motor bearings. The air gap between the rotor and the stator faces of modern motors is so small that slight wear on the bearings allows the rotor to rub and cause considerable damage. The extremely slow wear of antifriction bearings virtually eliminates this trouble and consequent expense.

These bearings are also commonly used on the pinion shafts of grinding mills and on the bearings of vertical spindle-flotation machines. At Cananea the 6- by 12-foot rod mills have been supported successfully on a roller chain, each chain having 6 rollers mounted on roller bearings.

POWER

Power at reasonable cost is essential to commercial concentrator operation. The average power consumption at 10 copper concentrators in the Southwest is 14.75 kw.-hr. per ton of ore milled. Power is generated by four general methods: Oil-Diesel, oil-steam, gas-steam, and hydroelectric. The use of gas is limited to plants that consume enough power to justify a pipe line from the distributing centers. The use of hydroelectric power is limited to a comparatively small area, as there is not enough power from this source to supply all the needs. Fuel oil burned under boilers or in Diesel engines has been the most economical unrestricted source of power in the Southwest.

Steam Turbines

Steam turbines are the most important prime movers where steam is used to generate power. One advantage in steam power is the flexibility in regard to

kinds of fuel used, since fuel oil, gas, or coal may be burned.

Higher pressures and superheating in steam turbine plants have resulted in lower unit power costs because of higher thermal efficiency. The older steam-turbine plants used steam at 150 to 175 pounds gage pressure superheated to a final temperature of 450° to 500°. The modern high-pressure plants in the Southwest operate at 350 to 400 pounds gage pressure, superheating to a final temperature of 700°.

The data that follow show typical, direct unit-power costs for 3 turbine plants of different capacities at 90-percent load factor and with fuel oil costing $2 per barrel.

TABLE 1. - Power costs for steam-turbine units

Size of power unit	kw.	1,000	2,000	10,000
Labor	cent	0.155	0.083	0.020
Supplies	do.	.150	.100	.080
Miscellaneous	do.	.050	.045	.040
Fuel	do.[1]	.580	.540	.480
Total cost per kw.-hr.		.935	.768	.620

1 To correct for fuel oil at any other price allow 0.024 cent per kilowatt-hour for each 10-cent variation in the price per barrel.

The installation cost for units as large as 10,000 kw. is about $100 per kilowatt for a complete plant with auxiliaries; the cost is somewhat higher for smaller installations.

Steam pressures up to 1,200 pounds per square inch are used in modern turbine plants; economy, however, is greater in units of large capacity.

Diesel Engines

In the Southwest where oil is the most economic fuel the cost of generating power with Diesel engines is appreciably lower than the cost by other methods. For example, a concentrator having a capacity of 1,200 tons and requiring 15 kw.-hr. per ton of ore milled would require a 1,000-kw. power plant based on 75 percent efficiency. The relative costs of generating this power would be 0.935 and 0.683 cent per kilowatt-hour for steam turbine and Diesel units, respectively (tables 1 and 2). The saving in power with the Diesel unit would be 0.250 cent per kilowatt-hour or $16,000 per year. For a larger plant the saving per unit of power would not be as high; a considerable total yearly saving, however, should be apparent for a plant having a capacity of 10,000 kw.

For milling plants of 50 to 500 tons daily capacity, requiring power plants of 50 to 300 kw. capacity, the Diesel engine is more economical than any other prime mover where oil as fuel is chosen on a heat-cost basis. These engines show very little drop in fuel efficiency down to one half load.

TABLE 2. - Estimated direct costs of Diesel-electric power based on fuel oil at $2 per barrel and 90-percent load factor

Number of power units	1	1	1	1	1	1	2
Rating .... kw. ...	75	150	200	500	1,000	1,500	3,000
Power for auxiliaries do. ...	7	10	12	14	25	35	65
Ave. load factor .... percent	90	90	90	90	90	90	90
Total power generated kw. ...	1,620	3,240	4,320	10,800	21,600	32,400	64,800
Net power distributed do. ...	1,470	3,000	4,032	10,464	21,000	31,560	63,240
Chief engineer	0	0	0	0	0	1	1
Head operator	0	1	1	1	1	0	1
Shift operators	3	2	2	2	3	3	3
Repairmen	1	1	1	1	2	2	3
Oilers	0	0	0	0	1	2	3
Power generated per bbl. fuel oil ..... kw.-hr.	380	390	395	430	470	500	500
Operating labor per kw.-hr. ...... cents .	1.020	0.533	0.396	0.167	0.107	0.094	0.052
Repair labor .... do. ...	.340	.200	.149	.067	.048	.032	.030
Supplies .... do. ...	.204	.167	.149	.077	.048	.048	.048
Lubrication .... do. ...	.061	.040	.038	.025	.020	.018	.018
Miscellaneous .... do. ...	.025	.025	.025	.025	.022	.022	.022
Fuel oil 1/ .... do. ...	.597	.554	.542	.481	.438	.411	.411
Total ....do. ...	2.247	1.519	1.299	0.842	0.683	0.625	0.581

1/ For each 10-cent variation in price of fuel oil from $2 per barrel make correction of 0.022 cent per kilowatt-hour.

Estimated direct costs of Diesel-electric power based on 90 percent load factor and fuel oil at $2 per barrel are given in table 2. Fuel consumption of Diesel-electric power units of 75 to 1,500 kw. ratings is presented in figure 1, and unit power costs for Diesel-electric power are given in figure 2.

Figure 2 shows that for plants of less than 300 kw. capacity unit costs rise very abruptly because the operating and repair labor cannot be reduced in proportion to the power generated. For mills requiring less than 75 kw. the power unit should be installed in the mill so that the engine operator may divide his time between the engine and the mill machinery. In this event the power cost should not exceed 2 1/2 to 3 cents per kilowatt-hour with oil at $2 per barrel.

Motor Applications

For milling plants of 500 tons daily capacity and over the prevailing practice in the Southwest is to use alternating current because of the ease of transforming voltages and the fairly high potential practicable; a saving in the cost of conductors is thus effected. A.-c. induction motors are also preferred because they have no commutators and are therefore less affected by dust. Some of the mills in the Southwest are operated on 440-volt a.-c. current, but the arrangement most favored is to operate motors of 50-hp. or more on 2,200 volts and motors of less than 50-hp. on 220 volts. Motors as small as 25 hp. for 2,200 volts may now be purchased.

In concentrators where several squirrel-cage induction motors are used it has been found that some motors operate below full load even though they have been selected with great care. Because of the characteristics of this type of motor such conditions reduce the general power factor of the system. One of the favorite methods of power-factor correction in the Southwest is to use synchronous motors for some of the larger machines. These motors are usually specified for a given load at 80 percent loading power factor, thus allowing capacity to correct a certain portion of the general lagging power factor in the circuit. The following are typical examples of the application of synchronous motors in the Southwest.

Concentrator	Service
New Cornelia	Grinding mills.
Copper Queen	2 of 16 grinding mills.
Morenci	5 of 11 grinding mills and cone crushers.

The old hand-operated-type motor starter is almost obsolete in the Southwest, having been replaced by the remote-control push-button type. In most plants it is forbidden by the safety rules of the operating companies, since the remote-control type is much safer and also protects the motor from injury because of its automatic control of the motor acceleration. A.-c. motors of 10 hp. and less usually are equipped with a simple magnetic switch and are started at full voltage. Above this size the usual equipment includes autotransformers which apply reduced voltage until the motor speed is sufficient

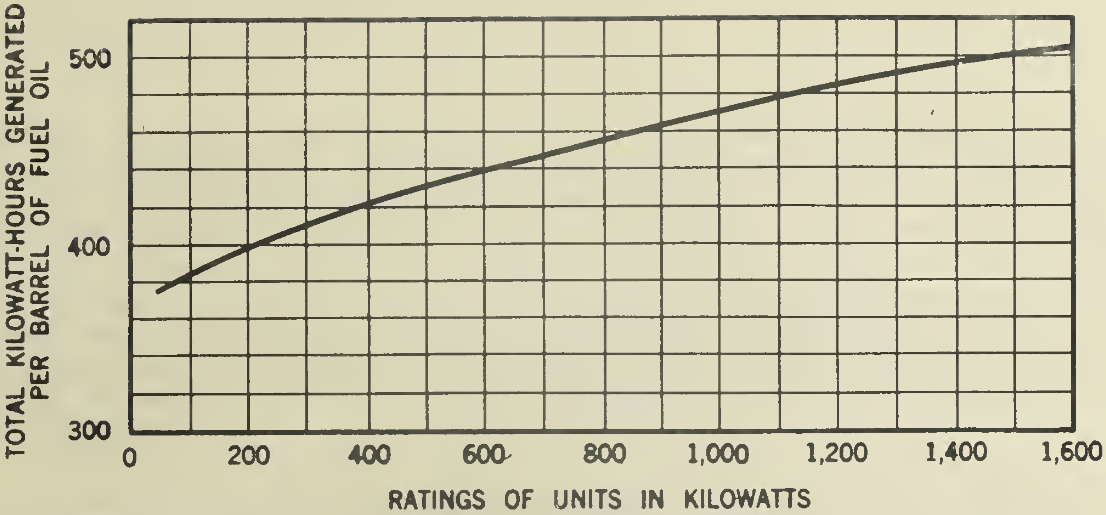

Figure 1.— Fuel consumption of Diesel-electric units, 75 to 1,500 kw. ratings.

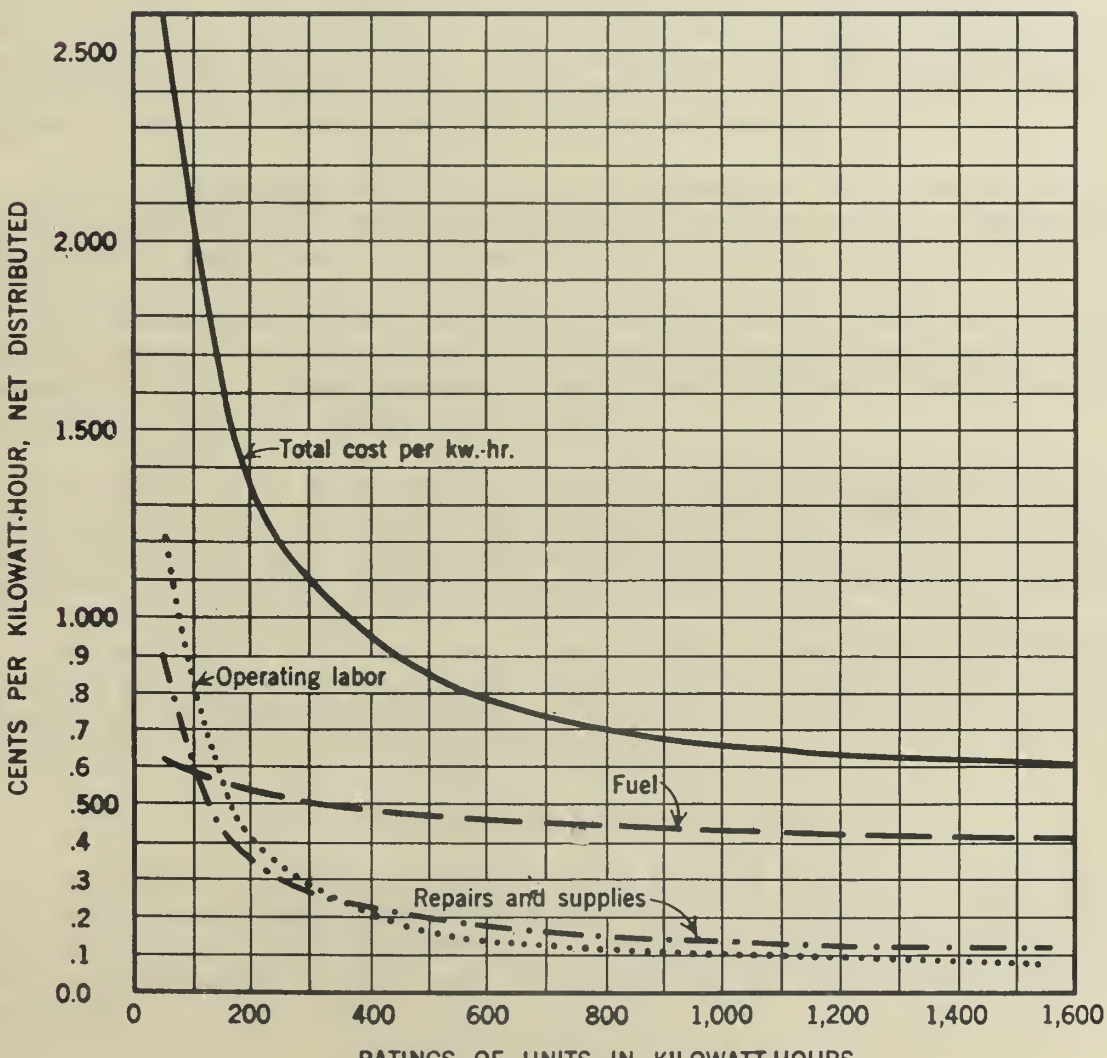

Figure 2.— Unit power costs for Diesel-electric units.

for full voltage. Synchronous motors have an induction winding and are started as induction motors; when full speed is reached the field circuit is closed, the motor immediately takes on synchronous speed, and the induction windings become inactive. The line-start motor is a comparatively new design; the starting equipment throws full voltage on the motor when starting. The starting equipment is much cheaper, but the motors are not quite as efficient. In consequence of the popularity of the push-button starters the trend is to enclose all starting equipment in dusttight boxes, which aid materially in reducing the repair costs and protect the apparatus and concentrator employees from accidents.

Standard squirrel-cage induction motors have a starting torque of 1.5 to 1.75 times the normal running torque, which is usually sufficient for driving conveyors, centrifugal pumps, blowers, and flotation machines. Where greater starting torque is required the wound-rotor-induction motor, also called slip-ring, is used. These motors can be obtained with a starting torque of 2.5 times the normal running torque. They are used to drive crushing rolls, grinding mills, heavy apron feeders, etc. Motors of this type are also used successfully where variable speed is desired, provided the machine driven requires a constant torque throughout the speed range.

D.-c. shunt-wound motors with field rheostat control are the most satisfactory where variable-speed conditions are exacting, as the relation of definite speeds to adjustment positions of the control is extremely reliable.

For wet or dusty conditions fully enclosed a.-c. motors may be employed. The motor itself is enclosed in a tight casing which is enclosed in another casing with an air space between. Fan blades circulate a current of air through this space to supply ventilation.

For small mills of 50 to 200 tons daily capacity where electric or partial electric drive is desired the d.-c. system is adaptable, especially in connection with Diesel power. As the logical location for the power equipment is at or close to the mill, transformation of voltage is not required.

An economical first-cost arrangement with good operating conditions is to drive some convenient equipment direct from the engine and generate enough d.-c. power at 110 or 220 volts to drive the rest of the machinery. Lighting may then be obtained from the power circuit.

COSTS OF CONCENTRATOR BUILDINGS

Geographical location - reflected in freight charges - and the probable life of the mine and concentrator are factors to be considered in choosing between wood and steel construction. In general, the cost of structural steel has been decreasing and the cost of timber has been increasing during the last 10 years.

Comparative costs per cubic foot of building for average conditions in the Southwest are presented in the following table. Estimates for structural steel are based on steel at $140 per ton erected and timber-work at $80 per thousand board feet delivered. Designs considered are for conditions in the Southwest.

Comparative cost per cubic foot of mill building for steel and wooden-frame structures

	Steel frame and concrete floors, cents per cubic foot	Wooden frame and wooden floors, cents per cubic foot
Excavations	1.3	1.3
Foundations and walls	2.0	2.0
Floors	1.2	.7
Building frame	8.4	6.9
Corrugated steel roof and sides, windows	.6	.6
Total	13.5	11.5

In erecting concentrator buildings there are so many items with fixed costs that the saving by wood construction is a small percentage of the whole. In a building required for a concentrator of 1,000 tons capacity the net saving of wood over steel construction would amount to approximately $5,000 and for a concentrator of 100 tons capacity, about $800. Approximately 1 ton of steel frame will be required for each 1,660 cubic feet of building, or 1,000 board feet of timber frame will be required for each 1,160 cubic feet.

ADVANTAGES OF LARGE-CAPACITY UNITS

Initial Costs

The advantages of large-capacity units for concentrating plants have been demonstrated recently in the Southwest by the changes and substitutions made from time to time to increase capacity.

In one concentrator having a capacity of 4,000 tons per day the following represents the original cost exclusive of the power plant:

Excavations and concrete	$ 650,000
Buildings and structures	600,000
Equipment	2,522,000
Total	$3,772,000
Cost per ton of daily capacity	$ 943

Had larger ball mills, classifiers, and flotation machines been installed at the time of construction the daily capacity could have been 6,000 tons at an additional cost of $950,000. The total cost would have been $4,722,000

or $787 per ton daily capacity. In this particular instance, if a capacity of only 4,000 tons was desired, the saving gained by large-capacity units would be about $300,000 and the resulting cost per ton of daily capacity, $868.

The comparison that follows shows the general effect of large-capacity units on floor space. A mill section with a capacity of 500 tons contains two 6 1/2-foot-diameter grinding mills carrying a combined grinding-media load of 60,000 pounds. The dimensions of the section are 27 by 140 feet or a floor area of 3,780 square feet. The area per ton of daily grinding capacity is 75.68 square feet.

A design for a 1,000-ton section on the same class of ore has 2 grinding mills carrying a combined grinding-media load of 120,000 pounds. The dimensions of the section are 32 by 166 feet, which allow for the required increase in size of classifier and flotation equipment. The floor area per ton of daily capacity is 53.12 square feet. These figures indicate a saving of 29.52 percent of floor space per ton gained by doubling the capacity of the essential units of equipment.

Maintenance Costs

The number of moving parts in a concentrator for a given tonnage can be reduced materially through the use of high-capacity modern equipment. The number of repairs and replacements usually is proportionate to the number of machines used; although the cost of renewals is a little higher than for smaller units, it is not over 50 percent more than for a machine of half the capacity. A reduction is thus effected in the cost of labor and supplies for maintenance in the operation of large-capacity units. The cost of repair labor and supplies ranges from $0.04 to $0.10 per ton of ore milled for large and small concnetrators, respectively; the probable saving due to the use of large-capacity units alone is $0.01 to $0.03 per ton of ore treated.

Operating Labor Costs

Under conditions that allow enough margin operating labor costs can be reduced noticeably by the use of large-capacity units. For example, if a proposed concentrator is to be equipped with 3 grinding units which could be replaced by 2, then either arrangement would require 1 man per shift. However, if 12 grinding units, requiring 2 men per shift, could be replaced by 8 larger units, there would be a possible saving of 1 man per shift.

EFFECT OF PLANT CAPACITY ON COSTS

Initial Costs

The data that follow have been computed on the assumption that large-capacity units are used.

Nominal daily capacity, tons	Initial cost per ton daily capacity
20,000	$ 550
10,000	625
5,000	675
2,500	690
1,000	700
500	710

The cost of coarse-crushing and conveying equipment for large plants is much less per ton than for the smaller installations. Small plants require 1 coarse crusher and 1 intermediate machine; although larger plants require larger units, the cost per ton of capacity is less. Shop building, tools, assay office, and administration buildings for larger plants also cost much less per ton of capacity than for smaller plants.

Operating Costs

Plant capacity influences operating costs more than any other single item. The tabulations of costs shown in table 3 are typical of the Southwest but do not represent figures taken from particular plants. Power costs are based on 0.5 cent per kilowatt-hour, graded up to 0.7 cent for the small plants. Power consumption is assumed at 15 kw.-hr. per ton milled. New water is taken at 300 gallons per ton of ore milled and at a cost of 5 cents per thousand gallons. Labor and commodity prices are the average for 1930.

TABLE 3. - Typical unit operating costs for copper concentrators of the Southwest

Daily concentrator capacity, tons	20,000	10,000	5,000	2,500	1,000	500
Operating labor	$0.040	$0.070	$0.100	$0.140	$0.210	$0.250
Repair labor	.020	.038	.050	.070	.085	.100
General department labor	.015	.025	.035	.055	.080	.100
General department supplies	.005	.008	.012	.015	.027	.033
Operating supplies	.110	.110	.110	.111	.122	.123
Repair supplies	.010	.010	.010	.011	.011	.012
Sampling and assaying	.005	.008	.015	.020	.030	.035
Power	.075	.075	.075	.090	.100	.110
Water	.015	.015	.015	.015	.020	.025
Total	$0.295	$0.359	$0.422	$0.527	$0.685	$0.788

As shown in table 3, general department expense and operating labor contribute mainly to the higher operating costs for the smaller-capacity plants. Even the smallest plant shows administration expense, as it must have a superintendent, master mechanic, and chief chemist. Operating labor must be supplied for crushing, grinding, and concentrating; if the plant capacity was increased within certain limits, the operating labor provided for the smaller plant would be sufficient to operate the large unit.

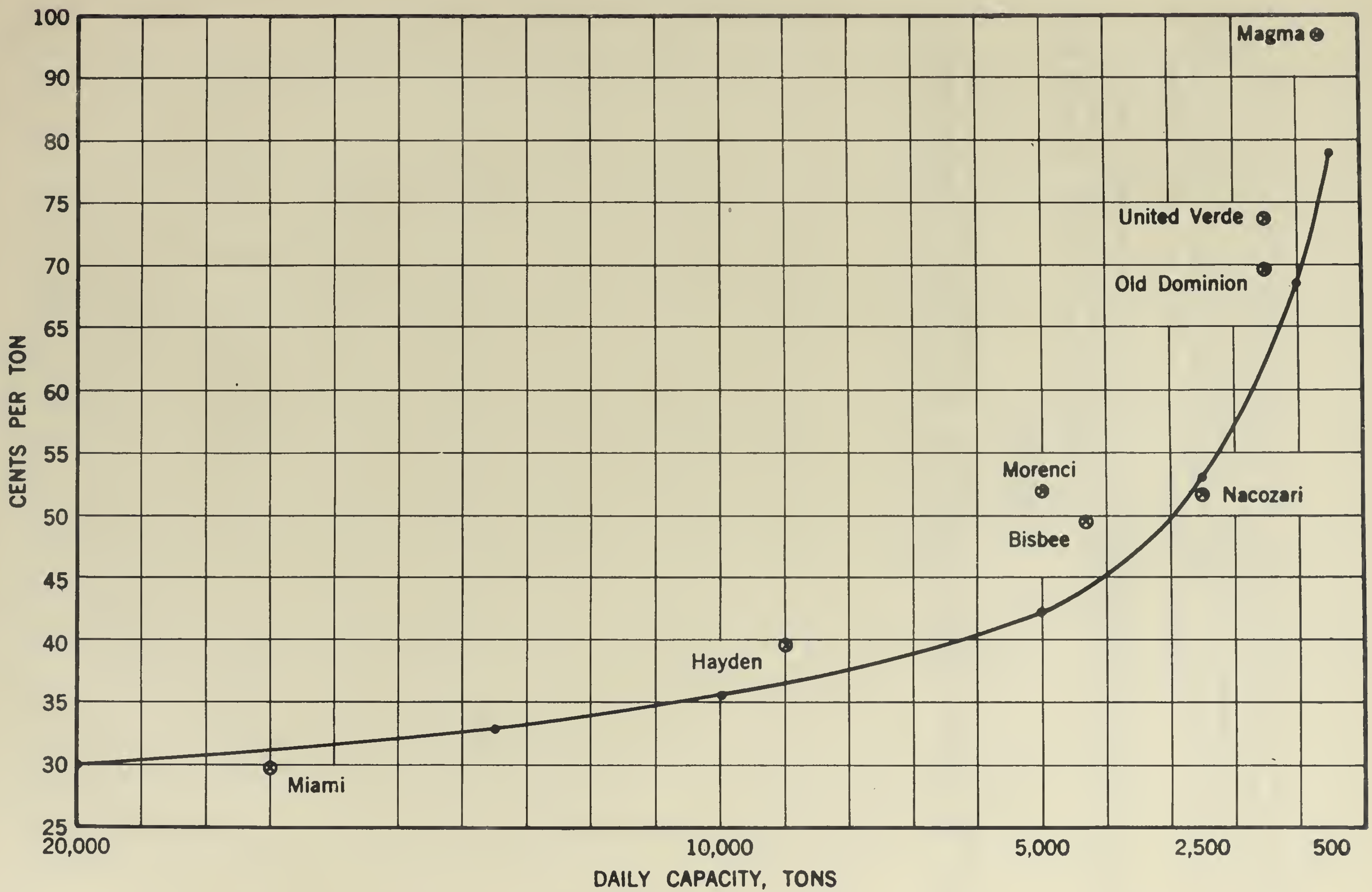

Figure 3.— Variation of unit milling costs with plant capacities and comparison with costs obtained at copper concentrators in the Southwest.

When units costs at the Southwestern Copper Concentrator are plotted against daily capacities the agreement is remarkably close, despite variation in the costs of the different elements. By making adjustments to suit prevailing prices, a curve can be obtained to show a very close approximation of expected operating costs at any given daily capacity.

A curve showing the variation of unit concentrating costs with plant capacities is given in figure 3. Typical southwestern operating costs are also shown in figure 3 for comparison.

CONCLUSION

The use of modern mechanical equipment in a well-designed assembly and of large-capacity units have been the means of lowering initial and operating costs at copper concentrators in the Southwest.

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INFORMATION CIRCULAR

SILICOSIS AS AFFECTING MINING WORKMEN AND OPERATIONS

BY

D. HARRINGTON

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

SILICOSIS AS AFFECTING MINING WORKMEN AND OPERATIONS[1/]

By D. Harrington[2/]

The occurrence of dusts in underground workings creates many difficult and, in some instances, contradictory and conflicting problems for workers, operators, and all others interested in health and safety in mining and tunneling. Moreover, there are many misconceptions both as to the actual effect of most dusts which may be encountered in underground work and as to remedial or alleviating measures. Although the harmfulness of mine dusts to health of miners has been mentioned by many writers in ancient times, much of our really authentic knowledge of the effect of underground dusts upon health or safety in both coal and metal mines has been gained during the past 30 or 35 years. There has been, however, a considerable amount of generalizing from some local condition or occurrence or theory rather than arriving at conclusions after extended studies and observations made not only in mines and tunnels but also in town, camps, and plants, and comprising data on as many different mining and tunneling localities, conditions, etc., as could be secured with reasonable expenditure of time or money. This paper gives some observations based on the writer's 30 odd years of experience in and around both coal and metal mines, including approximately 20 years' close study of dust disease with detailed surveys and reports on dust occurrence and effects in considerably more than 100 mines in probably 20 States of the Union. The observations are those of an engineer and in many respects conflict with the "going" ideas of doctors, pathologists, and others who have written on dust disease.

Where the word dust is used in this paper in connection with respiratory disease, it refers to solid particles which float or can float in the air breathed by workers. Although these particles are usually dry, they may be wet; in general, their dimensions are so small as to be invisible to the naked eye. Dusts in coal mines which may be involved in the initiation or extension of explosions, are generally thought to be those which are smaller than 20-mesh in size but chiefly finer than 100-mesh; coal-mine dusts which are harmful to health of workers have essentially the same general physical characteristics as other dusts, but usually consist of smaller particles.

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From the writer's personal investigations and observations, as well as from fairly close study and contacts of various kinds with the investigations and observations of others, there appears to be warrant for the conclusion that any dust, or combination of dusts - whether in a coal mine, a metal mine, or a tunnel - which is insoluble or difficultly soluble in the fluids and tissues of the respiratory organs is likely to be harmful ultimately to the health of underground workers if it is present in the air in minute form and is breathed in large quantity over large portions of the working shift for any considerable number of months consecutively per year. However, some soluble dusts which occur in underground workings are also decidedly harmful. It appears that the quantity of dust likely to be found in the air of underground workings, together with its insolubility, governs its hygienic harmfulness much more than its specific physical or chemical qualities. This does not mean that air containing a large quantity of finely divided flint dust or similar hard, sharp, insoluble siliceous or other material is not likely to be more harmful than air containing a similar quantity of probably less inherently harmful dust, such as very fine limestone, coal, or shale. Breathing comparatively small amounts of finely divided free silica or similar hard, sharp, insoluble, supposedly more dangerous type of dust, is, however, likely to be much less harmful to health than breathing large amounts of less inherently harmful dust, such as that from coal, shale, hematite ore, limestone, or combinations of these. Moreover, the dust of free silica (probably the most harmful dust in air of underground workings) apparently does not always take a form equally harmful to all who breathe it.

Physical examinations of underground workers show that men who work in very dusty places in coal mines have definite respiratory disease, whether the dust consists of coal only or a mixture of coal with rock or shale or clay. Men have worked in coal mines and have then gone to work in dusty metal mines or tunnels and have become affected by miners' consumption, or have worked in metal mines or tunnels and later have been seriously afflicted with respiratory disease after having worked in dusty places in coal mines. While dusts that are insoluble or difficultly soluble in the fluids and tissues of the respiratory organs are chiefly responsible for respiratory disease in miners, there are certain soluble dusts of lead, arsenic, and rocks or ores which are definitely poisonous or which result in material harm to the health of workers. Many, probably most, present-day authorities on dust disease think that silica dust is harmful chiefly because, in the very finely divided form in which it enters the lungs, it goes into solution and causes attendant chemical reactions impairing or destroying the lung tissues.

The main sources of air dustiness in metal mines or tunnels are, in the order of their importance: Dry drilling of holes for blasting - the dry drilling of holes pointed upward, especially those from about 60° to vertical, is in general the worst producer of harmful dust in the air breathed by underground workers; blasting of dry rock or ore - which produces dangerous gases and throws immense quantities of finely divided dust into the air; shoveling, or mucking, of very fine, dry material at the working face, which is usually poorly ventilated; loading cars from chutes; dumping loaded cars into chutes;

and timbering. Dry crushing and other occupations in metal mines and in metal-mine mills also produce dangerously dusty conditions.

That the quantity of dust breathed is the important point in the harmfulness of dust is a certainty, yet to state what is that dangerous quantity or what is a safe limit is difficult - in fact, it is impossible - with our present facilities and knowledge. In South Africa an arbitrary limit of 5 mg, or 300,000,000 particles per cubic meter (or 300 particles per cubic centimeter) of air, was set. Recent reports, however, give the average air dustiness of South African mine working places as about 1 mg per cubic meter and less than 200,000,000 particles per cubic meter of air. There is probably not one dry mine or tunnel in the United States where the average air dustiness of working places is as low as 5 mg per cubic meter of air (the South African standard) or 10 mg per cubic meter of air (the standard set by Higgins and Lanza in their study of miners' consumption in the Joplin, Mo., district in 1915). A recent State regulation in the United States restricts the number of dust particles to a maximum of 15,000,000 per cubic foot of air or about 530,000,000 per cubic meter (somewhat more than 75 percent more than allowed in South Africa). In general, the average air dustiness of working places in our dry metal mines where dry drilling is done is above 20 mg per cubic meter of air. In many it is above 50. Dry drilling of upper holes in some cases throws into the air breathed by the driller as high as 7,000 mg of highly siliceous dust per cubic meter of air, or 1,400 times the maximum allowed in South Africa. As an average, dry drilling of the upper holes (those above 60°) in dry friable material, such as some kinds of granite or quartz, produces about 150 to 200 mg per cubic meter of air. With holes under 60°, dry drilling produces about 50 mg, while wet drilling makes about 5 to 20 mg of air-borne dust per cubic meter of air. The cuttings from dry drills in metal mines sometimes contain nearly 50 percent of dust that will pass through a 200-mesh screen, and a large percentage of this will probably pass through a screen of several hundred mesh.

While any dust present in large quantities and in finely divided form in the air breathed by underground workers undoubtedly is likely, in time, to be decidedly harmful to health, some dusts are more harmful than others, some individuals and some nationalities are more susceptible to harm than others, and some occupations or occupational conditions accentuate the harmfulness of the dust present. Other factors may contribute, such as high temperature or high humidity, or air unduly depleted of oxygen or high in such gases as carbon monoxide, carbon dioxide, etc., which may add to the dust harmfulness by depressing the worker's vitality or unduly accelerating his rate of breathing. Or again, the atmospheric conditions as to temperature, humidity, and freedom from gases may be ideal for performance of maximum quantity of work, but if the worker has an incentive through contract or piecework prices to work at maximum capacity, he may incur maximum harmfulness from the breathing of dust if it is present in the more or less still air in finely divided form and in large quantities. On the other hand, there may be good ventilation, with resultant currents of pure, cool, moving air, to alleviate the harmfulness of dust occurrence; or water may be used freely and efficiently with like good effect.

Insoluble or difficultly soluble dust in the underground air generally manifests its harmfulness to the health of miners as some form of lung disease. These various forms are given such names as miners' consumption, silicosis, anthracosis, miners' asthma, phthisis, and possibly pneumonia. In addition, dusts undoubtedly are largely responsible for much bronchitis, and some soluble dusts are poisonous. Dust disease, however, should be given the more inclusive and much more nearly accurate name "pneumoconiosis" or the simpler one "dust disease".

It is the general belief that the dusts most harmful to the lungs range in size from 0.25 micron, possibly as small as 0.1 micron, up to 10 microns (a micron being one one-thousandth of a millimeter or approximately one twenty-five thousandth of an inch). It is the writer's belief (though this is not in accordance with the general, almost universal, opinion of dust-health experts, especially the pathologists) that the harm is due chiefly to the filling or partial filling of lung cells, tissues, etc., with dust, thus preventing these tissues from performing their normal functions; and that there is additional harm from the continual irritation caused by these foreign materials and by the efforts made to dislodge them. Possibly, too - and this is the theory of most present-day dust-disease experts as to the harmfulness to the lungs in connection with the breathing of dusts - harm may be done by the chemical or other action which takes place when large numbers and amounts of the very fine dusts are dissolved in the lungs. That many of these small dust particles are dissolved is inferred from the fact that it has been proved that very small particles of flint and chert - free silica material soluble only with difficulty when the particles are of visible size - are definitely soluble in distilled water.

Dust particles which result in asbestosis and possibly in bronchitis and kindred diseases are probably larger in size - up to 50 or even 100 microns; these larger particles seem not to get into the lungs or at least not to remain there; but they cause considerable irritation in the respiratory passages, and also clog protective devices and prevent them from intercepting the finer dusts from going into the lungs. While the dust of free silica, such as quartz, flint, chert, etc., is probably the most harmful of the dusts ordinarily found in underground workings, physical examinations of underground employees who have worked in the dusty air containing coal and shale as well as in the dust from ores of hematite (iron oxide), calcite (limestone), and other essentially nonsiliceous material, show definite amounts of lung involvement, as indicated by X-ray photographs as well as by the usual physical or physiologic symptoms, but especially by shortness of breath.

Local conditions intensify the harmfulness of dust in the air. When men work on a contract basis they exert a maximum physical effort, hence breathe fast and deeply, and thereby draw maximum quantities of dust into the respiratory organs. Similarly, where the dusty air is high in carbon dioxide or low in oxygen respiration is accelerated, and the lungs take in a maximum amount of dust. When dusty air is stagnant with high temperature and humidity (for example above 80°F. and 90 percent relative humidity) the vitality of the body is depressed, respiration is accelerated, and the dust condition apparently gets its maximum opportunity to harm the worker. Very fine

dust (from 10 microns down), when once suspended in air by any mining operation, remains in suspension for long periods (generally several hours - even with material of fairly high specific gravity) and the worker breathes this dust-laden air practically throughout the shift, if the air is stagnant; hence it is important to have currents of fresh air continually sweeping through and past all working faces and places. Good ventilation (next to the efficient use of water) is probably the best preventive of dust disease in mines, but the fresh, pure air must be circulated where the men work.

Some individuals seem to be immune to dust disease; old-time underground workers (with underground experience of 30 or more years in dry, dusty mines) have told the writer about numerous friends and coworkers who succumbed to dust disease, yet they themselves appeared to be healthy. On the other hand, there are records of men who were in their graves from dust disease within a year of the date of first underground work; and serious dust involvement due to 2 or 3 years' underground work is not at all uncommon. Racially the Finns, the Irish, and the Negroes seem most susceptible to health harm from dust. Environment and living conditions unquestionably have a vital effect on occurrence of dust disease, and underground workers whose homes are hovels, whose eating and sleeping habits and accomodations are abnormal or subnormal, who are addicted to drink, who are afflicted with one or more of the various so-called social diseases, are very likely to be susceptible to dust disease and ultimately to succumb to it.

While there is absolutely no doubt that dust diseases are directly responsible for the death of several hundred underground workers annually in the United States and indirectly responsible for the death or disability of several thousand others, unfortunately exact figures are not available, partly because many doctors apparently do not know how to diagnose the disease but chiefly because in the regions most afflicted a concerted effort usually has been made toward minimizing publicity about the dangerous conditions. In fact, "pussyfooting" about dust disease not only in mining but in industry generally is now and has for many years been largely the rule, and this more than any other influence has caused the present-day "racket" in connection with dust disease.

Instead of recognizing the menace due to the diseases of various kinds associated with the breathing of dust and instead of taking advantage of the methods available for combating or preventing them - or at least making a study of the situation with a view to alleviating the harmful conditions and their terrible results - there appears to have been a concerted effort to hide the facts in most instances and to take as few measures as possible toward elimination of the diseases and of the conditions causing them. Miners, especially metal miners, whose health is most directly affected usually refuse to allow themselves to be examined physically previous to or during employment. They frequently oppose the use of wet drills - which probably more than anything else will remedy the dangerous dust condition - and in many instances they destroy or try to make ineffective the water attachments to drills. When water is provided to wet down the dust it is generally not used, except when the mine boss forces the worker to it. And

metal-mine workers only too frequently destroy or partly destroy ventilation equipment installed for their safety, comfort, and health. Mine operators or officials sometimes deny the existence of miners' consumption or other dust disease among employees even when they know or suspect its existence, yet frequently some of the mine bosses themselves are afflicted by such disease. Suggestions as to remedial methods or equipment - such as sprinkling or the use of water in drilling or on muck piles, or the use of fresh-air currents to remove dust - are held to be impracticable or are considered the dream of the theorist, even though these methods or devices are in thoroughly successful use inother mines or in other localities. While sometimes the objection to change is due to disinclination or inability to withstand the probable financial outlay, frequently the main obstacle is the fact that mine officials are of the reactionary type; the old-timer with years of experience and more or less limited education usually hates to do anything that he has not been accustomed to doing. He repels the idea that dust in his particular mine is or can be harmful and very often denies the possibility of having enough dust present to be harmful.

In the South African mines on the Rand more miners are employed than in all of the metal mines of the United States, and the problem of dust disease is not "side-stepped" as it is and has been in the United States. About 30 or 35 years ago a very complete health study was made, revealing the extreme seriousness of dust disease among the workers and resulting in the adoption of drastic laws and regulations, which apparently are rigidly enforced and which are said to have resulted in a marked decrease in the incidence of the disease and in the death rate from it. Mines with 1,000 or more men underground must have men especially detailed on dust and ventilation work, whereas only a few of the metal mines or tunnel operations in the United States even consider dust or ventilation. South African mines must have running water in not less than a 1-inch pipe within 50 feet of the working place, and with a minimum pressure of 30 pounds a square inch; extremely few of our metal mines, on the other hand, have water available near faces. South African mines must have water blasts and sprays at or near all working faces; few, if any, of our metal mines have them. All drilling in South Africa must be done with drills in which water flows through the drill into the hole and prevents formation of dust; many of our miners and mine officials insist that "it can't be done," especially in the drilling of upper holes. In South Africa all blasting must be done after the men are out of the mine, the region where blasting is done must be thoroughly wet down before and after blasting, and men may not enter until 30 minutes have elapsed after blasting; in the United States, generally blasting may be done at any time, no provision is made to wet places at any time before or after blasting, and men return to blasted places essentially when they see fit to do so. South African mine traveling-ways and ore or rock piles must be sprinkled sufficiently to prevent the presence of dust in the air; very few metal mines or tunneling operations in the United States do anything toward sprinkling, and most of the operators say it is unnecessary and impracticable. In South Africa old mine workings must be closed, air currents must be split, air dustiness and chemical composition must be determined periodically by sampling and analysis, shift bosses must make

daily records of the use of water against dust, and many other preventive regulations practically unknown in the United States are in effect and said to be strictly enforced. Carbon dioxide must not exceed 0.2 percent in working places, whereas in the United States many of our workers in underground places breathe air with a high proportion (2 percent or upward) of carbon dioxide. Carbon monoxide must be less than 0.01 percent; but some of our mines have 0.01 to 0.05 percent of carbon monoxide at working places during all or a large part of the working shift.

In South Africa over 80 percent of the workers are native Kaffirs, and the white men present, acting chiefly as overseers, rarely do the hard, dangerous work at the face; in the underground workings of metal mines in the United States, on the other hand, practically all work is done by white men where natural conditions frequently are even more unfavorable than are those of South Africa. We have very few and very incomplete regulations, and most of these are ignored.

The outstanding remedy for the bad situation with respect to dust disease is education - education of underground workers in the necessity for taking such precautions as are available; of employers in the importance of recognizing the seriousness of the situation, providing devices and methods to reduce or prevent the incidence of the disease, and, if necessary, forcing their adoption on the workers; of doctors in the need for correctly diagnosing disease, giving publicity to its prevalence, seriousness, and remedies if available, etc., and issuing death certificates religiously assigning dust disease as the cause where such is the case; and of merchants, newspapers, and other influences in the community in the rightness of trying to prevent the disease rather than hiding its existence.

As specific remedial measures for metal mines and tunnels, the following are suggested: Mechanical ventilation, with a definite person or persons in charge, should be adopted to force moving currents of fresh gas- and dust-free air to every place where men work - to remove or at least dilute dust, heat, and gases in the air breathed by workers. The use of water should be enforced in all drilling, in sprinkling ore and rock piles, and in wetting timbers, manways, haulageways, and every place where dust may be found. Some dust experts are now discussing the abolition of the use of water in connection with prevention of mine-air dustiness, but the writer believes that next to providing ventilation to remove or dilute dust in air, the careful, systematic use of water offers the best present-known method of combatting the health hazard from breathing dust. Where possible, all the blasting should be done after a shift; where this cannot be done there should be enforcement of strict regulations as to the wetting of the region of blasting both before and after the shots are fired, as to the removal of all explosive fumes by adequate air currents, and as to the prevention of entrance into a blasted place until all dust, fumes, etc., have been removed. Metal mines in Canada which have been doing much work in connection with air-dust abatement have found that blasting is by far the worst mining practice in filling the mine air with the fine dust likely to do the most harm to persons engaged

in underground work. There should by all means be strict physical examinations of underground workers before employment and at periods of not more than 6 months during employment, with prompt removal of the worker from dangerously dusty work should unfavorable physical symptoms be found.

The dust-health problem in mining is likely to be very much in evidence in the future, and our mining men will find it to their advantage to look into this subject and to take quick, drastic action or the so-called "racket" on dust disease will have such serious consequences as to put many mining companies out of business. There is no question that far too much stress has been laid on silica dust and silicosis, whereas emphasis should be placed on the very definite dangers to workers in breathing a large quantity of any kind of finely divided dust which accumulates or is likely to accumulate day by day in the worker's system. The resultant respiratory disease should be termed "pneumoconiosis" or simply "dust disease" rather than "silicosis" or "anthracosis", etc., since it is highly probable that no person has ever been incapacitated or has ever died from breathing any single dust but rather from a combination of dusts, accompanied by other contributory influences or conditions.

While it appears that all dust and combinations of dusts which are breathed into the respiratory passages in large quantities and over considerable periods of time are likely to be harmful to health, unquestionably some dusts are relatively more harmful than others. Some dusts affect the lungs, some the bronchial apparatus, some the eyes, some the septum of the nose, some the stomach, and some the skin. Some individuals have considerable resistance to the harmful effect of breathing dust, others are readily susceptible; some nationalities are likely to succumb quickly to dust disease, and others are much better able to resist its inroads on health. Having breathed one kind of dust does not aid any human being to resist any other kind. Tuberculosis may accompany the other ills due to breathing of silica dust, but tuberculosis does not always accompany or result from breathing of silica or any other dust; tuberculosis may occur in connection with the breathing of coal dust or other dusts, though it is not so likely to be found in connection with coal dust as with some other types.

Silica in a chunk of rock, whether as free silica (SiO_2) or as a silicate, is practically insoluble in water but in the form of very finely divided particles which float in air it is readily soluble even in distilled water. No human being knows what percentage of silica in a dust is likely to cause silicosis, whether 1 percent, 25 percent, 50 percent, or more, and silicosis (so-called) is alleged to have been caused by breathing dust with as little as 3 percent silica. No person knows what quantity of dust in air (silica or other dust) is likely to be dangerous, though numerous arbitrary standards of air dustiness have been given to the public, these standards ranging from as low as 250,000 particles per cubic foot of air to 15,000,000 particles. With the present numerous, one might almost say innumerable, uncertainties as to what causes dust disease, as to what dusts or combinations of dusts are harmful, and as to many other important phases of the subject, it seems

anything but good policy - at least at present - to write into laws rigid requirements as to maximum allowable dust counts or weights. Certainly any such requirements should be labeled "tentative" if made at all.

Present methods of determining air dustiness, whether by weight or by count of particles, are anything but accurate or dependable, and it is now fairly well established that with present dust sampling and analytical methods results are likely to vary several hundred percent in accordance with the personal equations of the individuals who do the work. While the X-ray is unquestionably useful in determining whether respiratory passages have suffered dust harm, it is improbable that any person can determine from the X-ray alone whether the person X-rayed has silicosis, anthracosis, or any other specific type of dust disease - generally no two X-ray experts will check each other as often as 5 cases out of 10 in making interpretations.

Practically every human being breathes silica dust, because 50 percent or more of the earth's crust is siliceous - and probably 15 percent or more is free silica - and all of us breathe the dust from fields, roads, etc. Hence it is, or should be, perfectly apparent that it is <u>quantity</u> of dust breathed, and this includes free silica as well as other dusts, that determines air-dust harmfulness. Dusty air with high temperatures (that is, over 80°F.) and high humidity is more likely to be harmful because of dust than similar air with less trying temperatures. Dust-laden air contaminated with gases, such as carbon dioxide (over one half of 1 percent) or carbon monoxide (over one hundredth of 1 percent) is more likely to cause ill health than would air with the same amount of dust but without such high content of extraneous gases.

Dust disease may incapacitate its victim, or may even kill him, in less than a year from the time he first works in the dusty place, but in general it takes a much longer period. It is said that a person may work in a dusty atmosphere for several years without apparent ill health yet succumb to dust disease several years after leaving the dusty work. Generally, when a worker has become afflicted with dust disease, it has been (at least up to the present) impossible to cure him so as to completely remove the ill effects. This is aptly expressed by a doctor in The American Stone Trade, August 1933, as follows: "No treatment for the cure of silicosis is known. We can't make stiff tissue soft again. Prevention is the only recourse." It is said that sickness causes at least eight times as much loss of time by workers as do industrial accidents, and one of the most prolific causes of sickness of industrial workers is the breathing of dust. Dust disease unquestionably is the scourge of the industrial worker of today, and this is especially the case with the metal miner.

The mining industry should recognize that any and all dusts breathed in large quantities for long periods are likely to be dangerous to the persons who breathe them; then available measures should be taken to prevent air dustiness, the most essential being:

1. Use water in drilling, in sprinkling surfaces, and in connection with all dust-making processes.

2. Use the best possible ventilation to dilute or sweep away any fine dust which may get into the air where people work.

3. Do no blasting during the working shift and see that the air of working places is cleared of dust after blasting and before men return to work.

4. Install the best available equipment to prevent dust formation and avoid as far as possible the use of dust-producing practices, processes, or devices.

5. Have every person in the organization undergo a real physical examination before entering upon duty and at stated intervals (not less than annually), possibly with removal from dusty air as soon as symptoms develop in the respiratory tract; however, it should be mentioned that some well-informed persons do not believe that persons in the incipient stages of dust disease should be barred from doing further work in mines.

6. All those really interested in the preservation of the health of workers should familiarize themselves with available data as to dust dangers; a limited bibliography on this subject has been issued recently by the United States Bureau of Mines in Information Circulars 6835, 6840, 6848, and 6857.

7. While there is now an enormous amount of litigation as to dust disease, chiefly as to the misnomer "silicosis", and while a vast amount of material has been written and published on the subject, definite, dependable, accurate knowledge about dust disease (its causation, prevention, or remedies, medical or legal) is woefully lacking, and there is probably no more fruitful field for concerted, unbiased, disinterested investigation than that in connection with the health harmfulness from dust.

8. In the absence of more definite knowledge about practically all phases of dust disease, common sense would appear to dictate that mining people should by all means place themselves on the safe side by trying to prevent dust formation and to eliminate it so far as possible, if it must be made. In fact, the following admonition taken from the Journal of the American Medical Association, issue of September 9, 1933, may well be heeded:

> "There are admittedly many difficulties in the accurate diagnosis of the varied forms of pneumoconiosis. * * * * In more than one industry today the most important slogan in the 'new deal', from the standpoint of welfare defined in terms of human health and comfort, would be Stop the Dust."

9. Legal phases of the health harm from breathing dust are as complicated and as lacking in definiteness as are most of the other phases of the subject and they are made almost hopeless when court trials are under way by the lack of really dependable knowledge in the medical fraternity as to the disease.

Wisconsin has had wide experience on occupational diseases and an article entitled, "Dust, Fumes, Vapors, and Gases" in Industrial Medicine, November 1932, gives much interesting information and in connection with court cases on silicosis (so-called) is the following: "There were no two physicians who could, or would, agree on a diagnosis of any individual case" and adding that in general these doctors "could not be classed as giving other than such as conformed to their honest opinion and belief."

A lawyer writing on "Silicosis - In Certain of Its Legal Aspects" in the October, 1932, number of Industrial Medicine, among many interesting comments, says:

> "Throughout this article 'silicosis' alone has been mentioned. Silicosis is supposed to be a disease of the lungs resulting from silica dust, and the use of the term would indicate that the only dust that is of concern to industry is silica dust. This, however, is erroneous.
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> "Because sickness from dust exposure is commonly referred to as silicosis, there are many industries having a dust hazard that are not cognizant of the fact and remain in ignorance of it until they are greeted with a summons. Subsequent investigation and a conference with counsel bring home the fact that there is some merit to the suit, and that there is also an industry hazard that is going to prove expensive. The check-up will also often reveal, where there is insurance, that because this hazard was not known to exist there is no coverage for it, and that the problem is thus one which the company's lawyers will have to cope with and which the company will have to pay for.
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> "Concluding, the writer wishes to emphasize the two propositions: First, that it is extremely important that silicosis and kindred sicknesses be taken out of the common law and made a part of the workmen's compensation acts of various States; and second, that later legislation should be well-considered and should have the benefit of thorough study and research by capable and skilled medical men as well as the attention of lawyers fully cognizant of the needs and with original ideas as to the proper remedy."

Other lawyers are opposed to the inclusion of dust disease in the compensation laws, indicating that the lawyers as well as the doctors, pathologists, and others supposed to have a knowledge of dust disease are in anything but accord as to facts, procedure, or policies.

CONCLUSION

"STOP THE DUST" should be not only the slogan but also one of the major health and safety efforts of mining employers and employees, including those engaged in working in all kinds of dry mines. Ventilation is one of the most effective alleviating agencies and the use of water is a close second. Certain types of up-to-date equipment serve as very important adjuncts in the reduction of dustiness of air breathed by workers, but devices or equipment which are effective for one industry may be dangerously inefficient in others; some types of respirators may be used to advantage under what might be termed desperate conditions, but they are by no means "cure-alls" for the avoidance of disease from dust.

Dry drilling, with simultaneous utilization of certain dust-removal systems, seems applicable to at least some surface operations but has not yet been proved successful in many (if any) mines, and it should be remembered that it is impracticable to use dry drilling in some types of rock. If water is not used in drilling many underground faces will be very dry, and the dust from blasting, shoveling, loading, timbering, and other work will be kept floating in the air in maximum quantities unless ventilation is more than ordinarily efficient. Unquestionably, however, the most essential feature in the avoidance of law suits, sickness, disease, and death due to the inhalation of dust in industry is the recognition that all types of finely divided dust which float in large quantities in the air breathed by workers are almost certain to be harmful. This means that essentially all mines which are not <u>wet</u> are likely to have many places with dangerously dusty air and that the dust-making processes should be reduced to a minimum. If production of dust cannot be prevented, the dust should be kept out of the air as much as possible by the use of water or some other suitable method; when the dust cannot be kept out of the air, the air should be removed at once from working places - or at least diluted by adequate, controlled ventilation.

At present the mining industry (as well as numerous other industries) is confronted with what is termed a "racket" in law-suits and other demands in connection with health harm (real or alleged) due to dust-in-air. There is good reason to believe that the admittedly bad plight of industry (including mining) in connection with dust disease is due to the fact that, up to and including the present time, engineering and operating officials have ignored or dodged the issue and left the matter in the hands of the medical, pathological, and legal professions. As a result, scores of millions of dollars in law suits over dust disease are now pending against industry (including mining) and very frequently industry has essentially no defense when the "show down" comes.

Health problems in the mining industry should be handled essentially as have safety problems. The doctor and at times the pathologist, possibly the texicologist, are needed in connection with accidents to determine the extent to which the victim has been injured and to try to save his life and limbs after an accident has occurred - or to restore him to health. But in

the prevention of accidents (or of injuries due to accidents) the doctor is and probably always will be of relatively minor aid and the problem is one which must be handled chiefly by the engineer or the supervisory officials. Under this set-up, decided progress has been made in the prevention of accidents during the past 10 to 20 years and more especially in the past 5 years; on the other hand, with occupational-disease matters, including dust disease, left almost wholly in the hands of the medical and legal "fraternities" so little progress has been made in getting adequate, definite knowledge of dust disease, its cause, remedy, or method of prevention, that industry (including mining) is now confronted with a decidedly serious problem.

It has been known for much more than 20 years that dust disease has been more or less prevalent in our mines and factories yet so little has been actually done towards its eradication that the present "racket" is causing what is almost a panicky feeling in the minds of our mining and industrial executives. Volumes of abstruse medical literature have been published on the subject of dust disease but these have done little or no good towards clearing up the vast amount of uncertainty in the medical ranks as to the disease, its diagnosis, remedies, or almost any other strictly medical phase of the subject. Certainly the mining man, whether worker or operator, has been given essentially nothing as to the protection of the worker, or the type of dust or dusts or combinations of them, or the amount or size of dust which is likely to be harmful, or the composition of the so-called harmful dust as distinct from the so-called negative or inhibitive dust, or almost a score of other phases of the problem which most certainly should be known before trying to promulgate rigid standards whether in the form of law or otherwise. In other words, while the engineers and mine and industrial operating officials have either ignored or dodged the issue - and of course have done but little in the mine or plant to prevent health trouble - the medical people, pathologists, and others (very few of whom ever saw the inside of a mine) have given practically no definite data as to causation of the disease or of its treatment; and right now the medical, pathological, and other experts on dust disease are almost as far apart as the poles as to fundamental facts and factors in connection with the disease. As far as preventing the disease is concerned, relatively few of the most prominent dust experts ever saw the inside of a mine, hence any so-called advice they may give as to dust-disease preventive measures for mining should be (and usually is) essentially worthless.

On the legal side of the matter, there is almost as much confusion, chaos, and uncertainty as on the medical side and each State has its own procedure (or rather lack of it). As to what can be done in connection with lawsuits or prospective lawsuits whether from the side of the plaintiff or the defendant, much depends on local laws, local industrial conditions, or even the personality of local judicial authorities. It is the exception rather than the rule when real justice is done in decisions on dust disease, because of the almost innumerable uncertainties - medical, legal, and otherwise - about the disease. To protect both the worker from possible dust disease and the employer from future lawsuits or heavy compensation charges from this disease, it seems advisable that all of the precautions

now thought applicable be taken; at least, this gives maximum available protection to both worker and employer. The employer should remember that for generations it has been axiomatic under the law that it is the duty of the employer to provide a reasonably safe place in which his employees can work (and it now seems that "safe" includes the idea of health also); in general, the employer is also required by law to supply relatively safe equipment and appliances. The obligations of the employee are not so definitely known but, in general, if the employee refuses to utilize the precautionary measures or equipment furnished him, the probability of securing damages in court in case of accident or ill health is much diminished, and also if the employee supinely accepts working conditions which are so palpably unsafe or unhealthful as to be evident to any reasonable person, this may bar him from being successful in court procedure. Certainly, violation of State law is likely to cause the violator to lose his probable rights when in court. However, the legal side of the question is too abstruse and too confused for the layman, especially an engineer, to try to understand or expound.

This matter of preventing ill health in mining and in industry is just as definitely an engineering-operating problem as accident prevention. When a mine accident occurs, all of us are much interested in knowing the extent of the injury (information which the doctor alone usually can give), but we are even more concerned as to the conditions in the mine which brought about the accident and as to what can be done to prevent a similar (possibly worse) accident to someone else. Here the doctor is of essentially no aid, and the engineer and the operating man have gone a long way towards learning and applying the solution. As a result, our accident rate in mining (coal as well as metal) has been falling year by year - somewhat slowly but nevertheless effectively. Essentially a similar situation exists as to health in industry and especially as to dust disease in mining. The more definitely the matter is kept in the hands of the engineers and operating officials, taking counsel of course, with medical, pathological, legal, and other experts who are working on the problem, the sooner will mining and other industries be able to correct - or at least materially improve - the present almost intolerable situation confronting them in occupational diseases and especially in dust disease.

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COAL-MINE FATALITIES IN KENTUCKY IN 1934

BY

JOSEPH F. DAVIES AND H. B. HUMPHREY

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DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

COAL-MINE FATALITIES IN KENTUCKY IN 1934[1]

By Joseph F. Davies[2] and H. B. Humphrey[3]

A practical way of preventing accidents is to determine the causes of those of frequent occurrence and provide in every possible way against their recurrence. The following summary compiled from reports on fatal accidents in the coal mines of Kentucky for 1934 gives data on some of the causes of accidents and suggests measures for their prevention. Similar summaries and reviews have been issued in recent years by the Bureau of Mines[4] for this and other States.

ACKNOWLEDGMENT

The information used in compiling this summary was obtained from the monthly reports issued by John F. Daniel, chief of the Kentucky Department of Mines and Minerals, and made available through his courtesy and cooperation.

FATALITIES

All fatal accidents reported in and about the coal mines of Kentucky for 1934 have been included in this review. Accident reports for Kentucky are published annually by the State Department of Mines and Minerals and for all States by the United States Bureau of Mines. A few accidents in domestic mines and in mines not under the supervision of the State Department of Mines and Minerals are not chargeable directly to the mining industry. The revised mining law, effective July 1, 1934, placed more of the small mines under the jurisdiction of the State inspectors. Supervision should promote safer

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6868".
2 District engineer, U. S. Bureau of Mines Safety Station, Norton, Va.
3 Assistant mining engineer, U. S. Bureau of Mines Safety Station, Norton, Va.
4 Davies, Joseph F., and Humphrey, H. B., Coal-Mine Fatalities in Kentucky in 1931, 1932, and 1933: Inf. Circ. 6809, Bureau of Mines, 1934, 12 pp.
Miller, A. U., Review of Illinois Coal-Mine Fatalities for 1933: Inf. Circ. 6813, Bureau of Mines, 1934, 35 pp.
Herbert, C. A., Review of Coal-Mine Fatalities in Indiana During the last 3 months of 1932 and the Calendar Year 1933: Inf. Circ. 6828, Bureau of Mines, 1935, 22 pp.

working practices, as the proportion of accidents from these mines is high for tonnage produced and men employed; this fact is noted in reports from other States in which small mines operate without supervision of the inspection department.

Tables 1 and 2 compare production and fatality rates of Kentucky mines with those of 7 other principal coal-mining States for 1933 and 1934. As in past years, Kentucky stands fourth in tonnage mined. Although Illinois mined more coal in 1934, Kentucky (as in 1933) was third in man-hours worked, probably because of greater use of mechanical mining in Illinois. Kentucky was also third in the number of fatalities; however, the preliminary rates for 1934 given in table 1 show Kentucky and West Virginia with the same accident rate on a man-hour basis. Virginia, with a serious explosion disaster, had a still higher rate. The final figures for 1933, included for comparison, show the best record for the whole coal-mining industry experienced since accurate records have been available. The tentative figures for 1934 show a less favorable fatal-accident record, which probably will be increased as missing reports are received. Pennsylvania, Ohio, and Illinois maintained the good record of the previous year; Alabama, West Virginia, and Kentucky showed increases in accident rates which were, however, not unfavorable when compared with the rates for 1931 and 1932. The average for the bituminous mines of the United States for 1934 will be much better than for any year except 1933, according to present figures. The figures in tables 1 and 2 were compiled by the Employment Statistics section of the Bureau of Mines through the courtesy of W. W. Adams, Supervising statistician; they give the accidents and tonnage reported up to March 1935. The figures for 1934, of course, are subject to later revision.

TABLE 1. - Fatality rates of principal coal-producing States, 1933-34. 1/

State	Fatalities per million tons		Fatalities per million man-hours	
	1934	1933	1934	1933
Pennsylvania (bit.)	1.78	1.82	0.94	0.96
Ohio	2.11	2.76	1.20	1.57
Alabama	3.44	2.63	1.27	.97
Illinois	2.13	2.06	1.60	1.54
West Virginia	2.86	2.68	1.84	1.72
Kentucky	3.07	2.77	1.84	1.65
Virginia	4.29	2.81	2.42	1.59
Indiana	(2)	2.76	(2)	2.61
United States Average (bit.)	2.59	2.50	1.53	1.48

1/ Figures for 1933 are revised and final; figures for 1934 are preliminary and subject to revision.

2/ Reports incomplete.

TABLE 2. - Tonnage, man-hours, and fatalities for principal coal-mining States, 1934

State	Tonnage 1934	Man-hours[1] 1934	Fatalities 1933	Fatalities 1934
Pennsylvania (bit.)	89,223,000	168,345,000	144	159
Ohio	20,842,000	36,694,000	54	44
Alabama.........	9,596,000	26,005,000	23	33
Illinois........	40,905,000	54,540,000	77	87
West Virginia...	98,190,000	152,706,000	253	281
Kentucky........	38,068,000	63,980,000	100	117
Virginia........	9,100,000	16,135,000	23	39
Indiana.........	14,820,000	15,699,000	38	(2)
United States Total (bit.)...	358,395,000	606,421,000	833	927

1 Man-hours for each State were estimated by using the 1933 production per man-hour per State against the estimated tons given above.

2 Reports incomplete.

CAUSES OF ACCIDENTS

The number of fatal accidents by causes in Kentucky and in the United States is shown in table 3. The percentage of the total due to each cause also is given to indicate which causes were chiefly responsible for fatal accidents.

Falls of roof and coal caused 58 fatalities in Kentucky in 1934, but due to the large number of haulage fatalities the slate falls were only 46.8 percent of the total. When the fatalities from this cause in mines not under State supervision are subtracted the number is about the same as for 1932 and 1933, the improvement since 1931 being retained. Fatalities from falls of slate averaged about 57.6 percent of the total for all States combined.

Underground haulage accidents caused 21.0 percent of the fatalities in the coal mines of Kentucky in 1934 compared with 15.8 percent for the preceding 3-year period, and 17.6 percent for the United States in 1934. This continuation of the increase in fatalities from this cause which was noted in 1933 is significant. Undoubtedly haulage accidents are far too frequent, and steps can and should be taken to correct the conditions responsible. These factors are discussed in the section on haulage accidents.

TABLE 3. - Fatalities by causes in coal mines of Kentucky and the United States, 1934

Cause of accident	Kentucky		United States	
	Number killed	Percent	Number killed	Percent
Underground:				
Falls of roof or coal	58	46.8	535	57.6
Haulage	26	21.0	161	17.3
Gas explosion	2	1.6	37	4.0
Fire	5	4.0	8	.9
Machinery			17	1.8
Electricity	9	7.2	56	6.0
Explosives	3	2.4	25	2.7
Coal bumps	8	6.5	8	.9
Miscellaneous			22	2.4
Total underground	111	89.5	869	93.6
Outside:				
Explosives	7	5.7		
Haulage	3	2.4		
Machinery	1	.8		
(Sunstroke) (Falling timber)	2	1.6		
Total surface	13	10.5	59	6.4
Grand total	124	100.0	928	100.00

TABLE 4. - Fatalities according to occupation in Kentucky coal mines, 1934

Occupation	Number killed	Percentage of total number of employees in each occupation[1]	Percentage killed of total employed in each occupation[1]
Loader or miner	61	60.0	0.24
Machineman and helper	14	7.0	.47
Motorman	5	5.0	.25
Coupler or trip rider	21	5.0	1.00
Driver	1	.5	.35
Trackman and timberman	6	2.0	.52
Shot firer and driller	3	2.0	.43
Laborer	2	3.0	.15
Foreman and assistant	5	3.0	.50
Surface crew and miscellaneous	6	10.0	.12
Total	124	97.5	.30

1/ Approximate.

Fires and explosions caused 7 deaths in Kentucky compared with 45 for the United States, or 5.6 percent of the fatal accidents against an approximate average of 4.8 percent for the country as a whole. The only comment is that all such accidents are preventable.[5]

Nine fatalities (7.2 percent of the total) resulted from electric shock from low trolley wires; the average for the United States was about 6 percent. These figures represent an increase for both Kentucky and the United States over the preceding 3 years.

Explosives caused 3 deaths underground or 2.4 percent of the total, which is about the same as that for all the States averaged.

Coal bumps caused 8 fatalities (6.5 percent of the total) by direct violence. This problem is proving as serious as had been anticipated and fully warrants the studies that are being made.

On the surface an inexcusable explosives disaster cost 6 lives and haulage accidents 3 more; other accidents from various causes brought the number killed on the surface to 13.

OCCUPATIONS

In an attempt to discover the weak spots in accident prevention, the percentage of workers killed in each occupation proves interesting and valuable. The fatalities are grouped according to occupation in table 4. The number employed in the various occupations was estimated from representative figures compiled by the authors from mines in different sections of the State. Although not exact, these ratios are considered close enough for comparison of the risks. The approximate percentage killed of the total number of employees in each occupation is shown in the last column.

The work of the coupler or trip rider was so unsafe under conditions prevailing in 1934 that twice as many were killed in relation to number employed as in any other occupation. The ratio was about 1:100, compared with an average for all of 3:1,000. For the preceding 3 years the average for trip riders was 4:1,000. The next most hazardous occupations were those of trackmen and timbermen and machinemen; the percentages killed were 0.52 and 0.47 respectively. For the preceding 3-year period the corresponding rates were 0.32 and 0.43.[6]

The same percentage (about 0.24 percent) of the loaders and miners was killed as in the preceding 3-year period.

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5 Harrington, D., and Fene, W. J., Coal-Mine Explosions and Fires in the United States During the Fiscal Year Ended June 30, 1934: Inf. Circ. 6819, Bureau of Mines, 1935, 16 pp.

6 Davies, Joseph F., and Humphrey, H. B., Coal-Mine Fatalities in Kentucky in 1931, 1932, and 1933: Inf. Circ. 6809, Bureau of Mines, 1934, 12 pp.

Other occupations with high fatality rates were: Shot firers and drillers, 0.43 percent; and firemen and bosses, 0.50 percent.

Motormen were on a par with loaders; the percentage killed was about 0.25.

Sixty-one percent of the loaders were killed by falls of slate, 7 percent by haulage, 13 percent by coal bumps, and 3 percent by electricity; about 20 percent of the trip riders were killed by falls of slate; 51 percent by haulage, and 5 percent by electric shock; 45 percent of the machinemen were killed from haulage or tramming machines, 20 percent from explosives, and 15 percent from electric shock.

About 0.12 percent of all surface workers and 0.30 percent of all surface and underground men were killed.

AGES AND EXPERIENCE

The ages and years of experience of those killed in and around coal mines were tabulated and averaged. The average age of the victims was again 34 years, the same as for the preceding 3-year period. Only 10 (8 percent) were under 21 years, and 16 (13 percent) were over 50.

Without data on the number of men of these ages employed, the conclusion to be reached from these figures is that age has little to do with accidents, but added years probably have not brought added safety since more old men were killed than boys. In an attempt to estimate the significance of this question, other factors to be considered are impaired physical ability with increasing age, rashness of youth or inadaptability of age, and the relative number of old or young in hazardous jobs.

The average experience of all the men killed was 13 years, the same as for the preceding 3-year period. Only about 1 percent had worked less than a year in the mines, whereas 25 percent had worked 20 to 40 years, which should have added to their ability to work safely. Apparently the kind of experience and supervision has much more to do with producing safe workmen than mere length of service.

DEPENDENTS AND NATIONALITY

The average number of dependents was about 2.6 for each death, a smaller number than in the preceding years. About 23 percent had no dependents, which is a much larger proportion than in past years; 12 percent had 6 to 9 each, considerably fewer than in the past 3 years; 65 percent had families of 1 to 5.

Tabulation of the victims by nationality shows that 86 percent were native white Americans, 11.5 percent negroes, and 25 percent foreign-born; this is a considerable increase in the ratio of native white Americans and may indicate an increased proportion of native white labor employed over that of the 3 preceding years. Without data on the number of each nationality employed and their occupational risks no significance can be attached to these figures as regards relative safety.

FATAL ACCIDENTS IN 1934

The conditions that contributed to the 124 fatal accidents listed in table 3 were studied and with the causes and a summary of the recommendations for avoidance of similar accidents are grouped in the following tables.

Falls of Roof

During the year only 1 fatal accident was reported from falling coal; 56 fatal accidents, causing 58 fatalities (47 percent of the total), were reported from falls of rock from the roof or brushed ribs. These accidents have been grouped according to cause in table 5.

TABLE 5. - Fatalities from falls of coal and slate in Kentucky, 1934

Cause	Number of accidents	Number killed
Lack of inspection or attention to loose slate	20	20
Lack of systematic timber requirements	14	15
Caught by slate after knocking timber	12	13
Incompetent men in small mines	4	4
Inadequate supervision (direct cause)	3	3
Fell under slate while pulling it down	2	2
Disregard of instructions	1	1
Total	56	58

1. The direct cause of 20 accidents with 1 death each was failure to inspect the roof before working or going under loose or doubtful roof. In some mines regular inspection was not customary or required, being left to the worker; in others certain general rules had been made, but regular observance was neglected. Again, the section or mine forman was assigned to the duty of inspecting the roof before or after the men started work, depending on when he came around, but no one was responsible for the safety of the place before men entered. A number of deaths resulted from falls on haulageways which were not inspected and had not been scaled or timbered in a way to make them safe for travel. Most of these accidents involved faulty supervision.

2. The next most frequent cause of accidents was lack of a systematic timbering system requiring a minimum amount of support in all places and the setting of additional timbers where needed. There were 14 accidents from this cause and 15 deaths from falls in places where the timbering was left to the judgment of the miner as to how and when the posts should be set or to the special directions of the foreman on visits at varying intervals. Such matters should not be left to the men having the least ability in the organization or to directions which cannot be given at the proper time or which may not be strictly observed. This is almost entirely a matter of proper and safe supervision.

3. In 12 instances rock fell when timbers were removed without using a timber puller or without setting other timbers to take the places of the ones removed, and 13 men were killed. Often machinemen were the victims of these accidents. Deaths from this and the two preceding causes increased over those occurring in 1933, in spite of the warning and efforts of the State Department of Mines and Minerals to stop the practice of removing timbers unless safe means were employed. In some instances the roof was tested and found drummy, but a chance was taken and the posts were removed.

4. In four instances in small, uninspected mines not under the State inspectors, inexperienced or careless men were caught by falls of roof in places which they knew were dangerous. These have been grouped separately because they occurred in small unregulated mines.

5. In three accidents inadequate or incompetent supervision was the direct as well as the general cause. One foreman allowed men to work in a place he had pronounced dangerous but failed to have it made safe; another section foreman allowed a man to work in a place without timbers, although the rules required certain timbers in each place; the third accident occurred in pillar work which required careful inspection and timbering but on the day of the accident had not been visited by an official.

6. In two instances men were caught by slate which they were taking down. One slipped and fell as the rock came loose, and the other took a chance and tried to pass under while the rock was being pulled. To reduce such accidents the supervising officials should make sure that dangerous jobs are handled only by experienced, careful men.

7. In one mine an experienced man pointed out a dangerous piece of roof to his foreman and was told to set three safety posts before doing anything else. The foreman went on, and the man put off setting the posts and was killed. Although the man deliberately disregarded orders, the foreman shares the responsibility because of his failure to stay and see that the place was made safe.

Place of fatal accidents from falls of slate is shown in table 6.

TABLE 6. - Place of fatal accidents from falls of slate, 1934

Place	Number of fatalities	Percentage of total fatalities from falls of slate
At working face:		
Room	27	46
Pillar	12	21
Entry	8	14
On roadway or entry	10	17
In room	1	2
Total	58	100

The preponderance of fatalities in rooms over pillar workings indicates the proportionately small amount of pillar work being done. Eighty-one percent of the fatalities occurred at the working face compared with 74 percent in 1933. Rock falls on haulageways totaled 18 percent in 1933 and 17 percent in 1934. The location of most of the accidents bears out the contention that supervision is the means of preventing most of these accidents.

TABLE 7. - Occupation of men killed by falls of slate, 1934

Occupation	Number killed	Occupation	Number killed
Loaders	37	Foremen and assistants ...	3
Machine crew	6	Laborer	1
Haulage crew	5	Shot firer	1
Trackmen and timbermen	5		
		Total	58

In view of the smaller number of men employed as machinemen, haulagemen, and bosses, the number killed shows that these occupations are more hazardous than loading. The machinemen as a whole have a "chance-taking" attitude which must be changed before this type of accident can be reduced to any great extent. The haulagemen were killed because of failure to inspect and make safe the roof over roadways. Trackmen and timbermen often had bad places in which to work, but virtually all of them could have saved their lives by inspecting the roof and being reasonably cautious.

Prevention of Accidents from Roof Falls

A few well-known safety measures would have prevented most of the fatal accidents from falls of roof. Observance of safety measures specifically required in the mining law would have saved a few lives, and adoption of previous special recommendations of the inspectors would have saved others.

According to the causes given in the reports and recommendations, the following ordinary safety measures would have prevented many of the fatalities:

1. Inspect roof frequently and regularly.

2. Allow no one to pass or work under doubtful roof, except as may be necessary to make it safe.

3. Establish a definite minimum requirement for setting timbers.

4. Before removing timbers set others to take their places. If they cannot be replaced test the roof before knocking a post out and in most instances use a prop puller, rail, or long pole.

5. Make supervision and discipline of first importance. If the cooperation of men who disregard instructions cannot be obtained they should not be allowed to work.

6. Scale or timber bad roof only by or in the presence of capable bosses or men especially qualified for this work.

Coal Bumps

The coal-bump hazard has grown more serious in recent years and is being studied carefully by the State Department of Mines and Minerals, the United States Bureau of Mines, and the safety department of the Harlan County Coal Operators' Association, cooperating with several of the operating companies most seriously affected. Eight deaths resulted directly from five coal bumps. Other fatal accidents were thought to be due in part to less severe bumps. Several Mines experienced bumps of all degrees of violence without fatalities because no one happened to be in the line of violence. An investigation to determine ways of avoiding coal bumps should be continued in the affected areas. Not only are the bump and its violence dangerous, but a disastrous explosion might result from ignition of the dust cloud stirred up and the gas liberated by the pulverizing of large amounts of coal.

Investigations show that bumps are due to sudden release of stresses in the overlying strata, which are of great depth in these mountain ridges and are often largely unsupported because of the sharp ridges and ravines. The room-and-pillar system of mining often creates conditions that cause bumps, particularly if a long straight pillar line is not carefully maintained at all times and no stumps or "faulted" areas are left behind. One great trouble has been that the main roof would not break but would arch over and concentrate weight on the pillar faces or fracture suddenly with a shock like that of an earthquake and crush exposed pillars with great violence.

In general, investigators agree that a system of mining that would avoid the danger of bumps should be planned and carried out carefully. Such systems are recommended by the State Department of Mines and Minerals and by engineers who made the studies for the United States Bureau of Mines and the Harlan County coal operators, as well as by others.7

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7 Jones, D. J., Wilder, H. M., and Maurice, J. F., Mountain Bump in the Coal Fields of Harlan County, Ky.: Kentucky State Dept. of Mines and Minerals, Geol. Div., ser. 6, Bull. 1.
Rice, G. S., Bumps in Coal Mines of the Cumberland Field, Kentucky and Virginia - Causes and Remedy: Rept. of Investigations 3267, Bureau of Mines, 1935, 36 pp.; Bumps in Coal Mines - Theories of Causes and Suggested Means of Prevention or of Minimizing Effects: Am. Inst. of Min. and Met. Eng., Tech. Pub. 609, 1935, 23 pp.; Occurrence of Bumps in the Springhill No. 2 Mine of the Dominion Coal Co., Ltd.: Nova Scotia Dept. of Public Works and Mines, 1924.
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Hudspeth, H. M., Forces Induced by Extraction of Coal: Trans. Inst. Min. Eng., vol. 85, 1933; Coll. Guard., vol. 146, 1933, pp. 541 and 582.
Colliery Guardian, Bumps in South Staffordshire Thick Coal: May 26, 1933, pp. 756-758.

Fatal Haulage Accidents

Haulage accidents caused 23.4 percent of the fatalities in Kentucky coal mines in 1934. The 29 fatalities are grouped in table 8 according to the direct causes chiefly responsible.

The marked increase in fatal haulage accidents has been due largely to failure of both management and employees to adjust safe working practices to changed working conditions. Haulage equipment and systems suitable to older methods and hours of work usually cannot be adapted safely to increased territory and work with less idle time, unless the following requirements are met:

1. Adequate equipment and time given to each part of the haul.

2. A regulated and supervised haulage system.

3. Discipline and strict compliance with needed safety regulations.

Increased haulage accidents for 1934 have been reported from other important coal-mining States. Officials making these reports believe that properly maintained equipment, system, and discipline must be required.

TABLE 8. - Fatalities from haulage accidents in Kentucky coal mines, 1934

Cause	Number of accidents	Number killed
Wrecks and collisions	7	7
Run over getting on or off trips or cage in motion	7	7
Crushed between top and locomotive - lack of clearance	6	6
Run over while riding or walking in front of pushed trip	5	5
Trying to couple cars on the "fly"	1	1
Leg crushed in locomotive gears	1	1
Caught between cars and timber	1	1
Run over while blockholing	1	1
Total	29	29

1. Wrecks and collisions caused seven fatalities. A loader in attempting to move a locomotive let it run away and was killed; an electrician operating a motor ran up a slate fall; a locomotive left the track on a curve because of excessive speed and caught a loader against the rib; a locomotive ran into part of a loaded trip which had broken loose and stopped in a swag; a coupler was caught between a wrecked car and timbers; and a surface worker was killed when a loaded car jumped the track at a tipple. Haulage equipment should be operated only by those previously certified for the job.

2. The dangerous but common practice of getting on and off moving trips resulted in seven fatalities in 1934, one of which was on a cage in a shaft. Two reasons for taking this chance of death or being crippled appear possible - to save the company's time or to save personal time and bother and ignore commonsense rules.

3. Six men were killed by being crushed between the top and cars or locomotives. In two instances the men were inexperienced or incompetent. No one should be allowed to act as coupler or trip rider unless previously certified for the job. Lack of clearance, requiring haulagemen to squeeze into places not intended for riding space, was a factor in nearly every instance. This lack of clearance has been tolerated so long that it has come to be regarded as usual practice or even a necessary condition; however, several mines in the district find it more efficient and economical, as well as safer, to brush the roof of haulageways in low coal.

4. Another common practice, long condemned as dangerous, is walking or riding in front of pushed trips. There were 5 fatal accidents from this cause in 1934.

5. The other fatal accidents, all from causes which have been proved and branded as dangerous, resulted from coupling cars on the fly, backpoling, coupling cars in a place where there is not enough side clearance, and unguarded gears in a locomotive.

TABLE 9. - Occupation of men killed in haulage accidents, 1934

Occupation	Number killed	Occupation	Number killed
Motormen	3	Trackmen	1
Couplers	14	Machinemen	3
Driver	1	Rock-Duster	1
Loader	4	Electrician	1
Laborer	1	Total	29

The fact that only 17 of the 29 men killed in haulage accidents were members of haulage crews indicates a need for more careful regulation of all haulage practice and selection and maintenance of personnel.

Fatalities From Electric Shock

There were 9 fatalities from electric shock in 1934, more than in any previous year. Eight of the 9 deaths were due to low unguarded trolley wires and 1 to contact with a live uninsulated part of a locomotive which would have had no current in it if it had been properly inspected and maintained.

Fatalities resulted from contacts with the trolley wire when: A machineman rerailing a truck straightened up under the wire; a miner got out of a man-trip on the wire side; a coal loader helping to move some motors at a sidetrack stood up in the deck and touched the wire; a coupler helping to repair a locomotive stood up under the wire; a mine superintendent straightened

up at a parting while operating a locomotive; a motorman touched the wire while replacing the pole; a cable hook caught on a hanger while nipping and a machineman was pulled into the wire; and a coupler touched the wire while crossing over bumpers to rerail a car.

In all instances the wire was low and unguarded; the accidents would not have happened if proper clearance or guards had been provided. The Mine Safety Board of the United States Bureau of Mines recommends guarding all wires less than 6½ feet above the rail. With greater top clearance the wire could be placed at least high enough not to be a constant menace, and it could be guarded at the most needed points - sidings and switches. These are also examples of fatalities that occurred when uncertified men were allowed to operate haulage equipment.

Need for First-Aid Training

Accounts of the deaths from electric shock show that 5 of the 9 men might have been saved had first aid been given promptly by men with ordinary training. Unnecessary loss of life is strong justification for the requirement in the revised mining law of Kentucky that all men working in and about mines shall have first-aid training. These fatalities occurred in mines where such training had not been given.

Fatalities from Explosives

During 1934 there were 3 fatalities underground from unsafe practices in the use of explosives. A boy working in a small, unventilated domestic mine was suffocated by black-powder fumes when he entered 3 hours after blasting. A driller and his helper were fatally burned by pellet powder ignited by a short circuit in an improperly insulated drill cable. The supply of powder was in an open box on the truck.

A machineman after leaving the mine was struck by a piece of iron when a blast was fired in a waste dump; he failed to seek cover when warned.

Six persons were fatally burned by an explosion of 15 kegs of black blasting powder in a small building used as storehouse and office of a small truck mine. The powder was ignited by a pistol shot during an argument. The storage of explosives in any but a properly constructed magazine was a violation of the mining law. The powder had been placed there that morning.

Black blasting powder was involved in 9 of the 10 fatalities; the Bureau of Mines recommends that only permissible explosives be used in coal mines; evidently the recommendation is a sensible one.

Gas Explosion and Fire

A machine crew cut through the face of a heading which was approaching an old mine without waiting for drill holes to be placed in advance as had been ordered; the gas released was ignited by open lights or by electric arc

from the machine. The machine runner died from afterdamp and the helper from burns 4 days later. There was no violence. In addition to the need of permissible equipment in gassy mines, some stronger precautions were needed to see that the drill holes were placed and the cutting through done with the necessary care.

A small mine fire started by blasting with pellet powder caused smoke and fumes which asphyxiated five men when the usual shift was allowed to enter the mine. Blasting should be done with permissible explosives only, and no one but properly organized fire-fighting crews should enter a mine in which there is fire until the fire has been sealed or extinguished and normal ventilation restored.

Other Surface Fatalities

A carpenter was caught in a conveyor belt and pulley when he attempted to apply belt dressing with the belt in motion. A mechanic died of sunstroke while working on a fan engine outside the fan house; apparently his life might have been saved if first aid had been given. A mine official was killed by a falling timber while a tipple was being repaired.

CONCLUSIONS

The accident record of the coal mines of Kentucky did not improve during 1934, as it might have done had there been the same amount of attention given to safety regulations as in the preceding year. Even if the natural hazards - low roof clearance and bumping pillars - are not changed, insistence on precautions in inspecting and timbering doubtful roof and regulation and discipline in haulage and in electrical installations and usage will go far toward reducing the loss of life which has occurred year after year.

Cooperation and compliance with the present mining law will do much to bring about safer working standards. This cooperation is necessary from all within the industry and those dependent upon it, including mine managers and officials, mine employees, and all others concerned in the welfare of the coal miners.

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ASBESTOS - MILLING, MARKETING, AND FABRICATION

BY

Oliver Bowles

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December 1935

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DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

ASBESTOS - MILLING, MARKETING, AND FABRICATION[1/]

By Oliver Bowles[2/]

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1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6869.

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INTRODUCTION

This paper is the third of a series of reports on asbestos prepared by the Bureau of Mines. The reports already issued contain general information, including descriptions of deposits throughout the world.[3] The present paper deals chiefly with milling and marketing. Many grades of asbestos are marketed, and an important phase of milling is the grading of fiber; these subjects therefore are related so intimately that contiguity in their treatment is fully justified. Although the Bureau of Mines is interested primarily in the mining and preparation of raw materials, the fabrication of asbestos products has attained such importance in the United States that brief reference to processes of manufacture is included.

CLASSIFICATION OF FIBER

Canadian asbestos fibers are divided into three main groups - crudes, mill fibers, and shorts. The term "crude" is applied to fiber of spinning grade measuring three eighths inch or longer, that is hand-cobbed and not passed through a mill. Mill fibers are obtained by crushing and beating the fiber-bearing rock until the asbestos is freed and then removing the fiber from the rock by screening or air separation. Shorts are the lowest grades of mill products.

Russian asbestos also is classed as crudes and mill fibers, but the term "crude" is applied somewhat loosely; it apparently includes some spinning fiber that has been prepared mechanically.

This term is not generally applied to African products. Spinning fibers comparable in quality with Canadian crudes are prepared by hand-cobbing alone or in conjunction with simple mechanical crushing, disintegrating, and screening processes. A much larger proportion of the African long fibers have been prepared mechanically during recent years. Mill fibers of the shorter grades constitute the largest part of the tonnage.

PREPARATION OF FIBER FOR MARKET

General Features of Milling Methods

As pointed out in an earlier report the two major types of chrysotile occurrences are (1) deposits derived from alteration of peridotite-pyroxenite rocks and (2) deposits derived from dolomitic limestone intruded by diabase. The first type is the principal source of asbestos in Vermont, Quebec, the Ural Mountains, Rhodesia, and the Transvaal. The second type, which has little commercial value, occurs in Arizona; Minusinsk, Siberia; and the Carolina

3/ Bowles, Oliver, Asbestos - Domestic and Foreign Deposits: Inf. Circ. 6790, Bureau of Mines, 1934, 24 pp.; Asbestos - General Information: Inf. Circ. 6817, Bureau of Mines, 1935, 21 pp.

district of South Africa. The principal gangue material associated with asbestos of the first type is serpentine, and the materials with the second type are serpentine and limestone. As chrysotile also is serpentine, differing only in structure from the gangue, it is obvious that the milling of chrysotile must differ from all other forms of ore concentration. As the valuable mineral and the gangue consist of the same material, neither chemical composition nor specific gravity can be utilized as the basis for separation. The property that makes mechanical separation of chrysotile possible is its fibrous structure, which permits it to be opened or divided by impact into separate filaments, thus making it amenable to separation from the gangue by air suction and screening.

The value of chrysotile asbestos depends largely on the length of the fiber. When fiber three eighths inch long is worth $100 a ton fiber three fourths inch long is worth approximately $400 a ton. Thus it is apparent that breaking fibers in half destroys approximately three fourths of their value. The most important principle underlying the milling of asbestos therefore is the separation of fiber from rock with minimum breakage of fibers. The best machines spare the fiber unnecessarily rough treatment and at the same time break the rock effectively. Modern mills are designed to remove the separated fiber after each crushing process, to separate the sand as soon as it is formed, and to keep hard, barren rock out of the mill. If fiber already freed from rock enters the next crushing unit along with sand and rock fragments, it will be broken into shorter grades. Asbestos milling consists essentially of coarse crushing, drying, and recrushing in stages, each step being followed by screening and air separation of fiber from rock.

Another peculiarity of asbestos milling is the absence of any known method of testing the quality of mill feed. A rock that appears to contain little fiber may yield a fair return; conversely, one that appears promising may show a disappointingly low recovery. As both the valuable mineral and the gangue are serpentine an assay is useless. A laboratory-scale crushing and fiber-separating unit might give results that differ widely from those of a commercial mill operating on the same rock. Although the percentage of recoverable fiber in a rock cannot be determined in advance the actual yield can be calculated from the mill output.

In the following discussion of milling methods throughout the world Canadian practice is considered first, because Canadians were the pioneers in the asbestos industry, and their methods have established a basis for practice in other countries.

Canada

Preparation of Crudes

In Canada the associated rock is broken free of the larger-fiber veins by sledging, and the masses of fiber are sorted by hand. The fiber with some adhering rock is dried on steam coils and then taken to the cobbing sheds where the fiber veins are flattened with a hand hammer, freeing them from adhering

rock. The separated rock, dust, and short fibers are dropped into a receptacle under the bench, and the remaining high-grade fibers are classed as Crude No. 1, material 3/4 inch long and over, and Crude No. 2, material 3/8 to 3/4 inch long. To separate further the short fiber and rock dust, Crude No. 1 is screened on a flat shaking screen with 3/8-inch holes and No. 2 on one with 3/16-inch holes. After being screened the fibers are bagged in jute bags of 100 pounds capacity. Five to twenty percent of the rock fragments, dust, and short fiber still remain in the crudes as sold, and these impurities are separated at the factory where asbestos products are manufactured. Rejects accumulated in preparing crudes are utilized as mill fibers or sold as screenings. Crudes constitute a small proportion of the recoverable fiber but are the most valuable products.

Ru Keyser has developed a process of preparing long fiber mechanically, by which rolls are used instead of hand hammers.[4] The fibers are graded on screens which are so operated that the fibers are held in a horizontal plane. The claim is made that fibers thus prepared are superior in grade and purity to hand-cobbed crudes.

Milling Practice

Primary crushing. - Rock from the quarries or mines is dumped into a so-called "sluice" that holds several carloads. The sluice may have a railroad-rail-grizzly bottom which provides a bypass for fines. The primary-crusher feed from the sluice is controlled by finger gates, that is, suspended rails which may be moved up or down. A jaw crusher with a 36-inch by 24-inch opening set for a 4- to 6-inch discharge is the most popular primary breaker. Larger or smaller units may be employed as needed.

Sorting. - The crusher discharges to a picking belt to remove liberated crudes.

Drying. - The quarry rock contains considerable moisture, in winter even snow and ice, and it must be dried for proper milling. Standard rotary driers generally are used. The shells are 40 to 60 feet long, 4 to 6 feet in diameter, and inclined at an angle of 7°. The material is lifted by a series of blades and allowed to fall through the current of hot air that circulates through the cylinder. Capacities range from 30 to 60 tons of rock an hour and costs from 4 to 18 cents a ton.

Stack driers also are used and said to be much more economical. Rock is elevated to the top of a 50-foot stack 7 feet in diameter and allowed to fall through a rising current of hot air. Grid bars interrupt the speed of the descent.

Storage. - A large supply of rock in storage insures regular mill feed. Rock is fed to the mill on belt conveyors carried in concrete tunnels beneath

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4/ Ru Keyser, Walter A., Mechanical Cobbing of Chrysotile Asbestos: Eng. and Min. Jour., vol. 134, no. 6, June 1933, pp. 235-237.

the storage bins. By drawing rock from several gates at the same time material of comparatively uniform size and fiber content is obtained. Bins having capacities of 25,000 to 150,000 tons are in use.

Fiberizing, screening, and air separation.- Although crushing, drying, and storing methods are fairly uniform throughout the Canadian district milling methods vary greatly. No flow sheet can be applied generally because milling must be modified to suit the rock. A method suitable for hard rock would be too severe for fiber enclosed in soft rock. Secondary crushing may be done with jaw or gyratory crushers, rolls, Symons cones, or hammer mills. A straight impact that frees and fluffs the fiber is preferred to a grinding or shearing action. Rolls therefore are less desirable than impact mills.

The following flow sheet given by Ross[5] probably represents the best modern practice.

Grizzly in rock sluice
Jaw crusher
Drier
Grizzly, fines to screens fitted with suction
Gyratory crusher
Screens with suction
Rolls
Screens with suction
Jumbo
Screens with suction
Jumbo
Screens with suction
Jumbo
Screens with suction
Jumbo
Screens with suction
Rotary graders and flat cleaning screens
Floats bin
Sand to dump

It will be observed that the principle of removing fiber after each stage of crushing is followed strictly.

According to this flow sheet crushing beyond the primary stage is accomplished with gyratory crushers and Jumbo machines. The Jumbo, which is manufactured locally, is a horizontal drum 6 to 8 feet long and 24 to 30 inches in diameter. Beaters attached to a horizontal shaft consist of arms that support manganese-steel hammers. The beaters are set at intervals of 6 or 8 inches along the shaft and clear the inside of the shell by about 1/2 to 1 inch. They are turned at an angle to direct the broken rock toward the discharge end. The shaft rotates at 400 to 800 r.p.m., according to the nature

5/ Ross, J. G., Chrysotile Asbestos in Canada: Canada Dept. of Mines Bull. 707, 1931, p. 42

of the rock treated. Instead of the Jumbo machine some mills employ various types of cyclones, which beat the rock to a powder. The Laurie cyclone consists of a cast-iron chamber in which two beaters shaped like ship propellers are driven in opposite directions at a speed of 1,700 to 2,000 r.p.m. In the Pharo cyclone the beaters revolve in the same direction, and the rate of discharge is controlled by gates. The Torrey cyclone is somewhat like the Jumbo, except that the axis of the beater arms is vertical.

Shaking screens generally are used, although vibrating screens of the Hum-mer type also are employed. Fines pass through the screen, but the coarser rock fragments and longer fiber remain on the screen. The shaking action brings the fiber to the surface of the bedded-rock fragments; at the discharge end of each screen the fiber is lifted and removed by suction. The opening of the suction hood is 3 to 4 inches wide and extends across the full width of the screen. Thus, each screen furnishes three products - fines that pass through as waste, coarse rock fragments discharged at the end, and fiber collected by the suction fan. Oversize fragments pass to the next fiberizing unit.

The fiber is carried through pipes to a collecting chamber, where the diminished rate of the air current permits it to fall. The dust-laden air from the fans is blown into chambers, where the fine fibers are collected as "floats."

Wet process. - Considerable progress has been made in developing a wet process for treating Canadian asbestos. Some claim that wet treatment involves less breakage of fiber and a higher recovery. Experiments were begun in 1921, and a testing plant was built in 1923, but commercial usage has not been attempted.

Grading. - Fiber usually is graded according to length in trommels 3 to 5 feet in diameter and 5 feet long fitted with woven-wire cloth. A central shaft is equipped with wooden paddles that agitate the fiber and force the shorter grades through the screen mesh. One type is a stationary trommel in which the paddles rotate at 600 to 800 r.p.m. Another type rotates in one direction at 5 r.p.m., and the paddles rotate in the opposite direction at 180 to 250 r.p.m. Some trommels make only one finished grade - that passing through the screen - and the oversize is conducted to a second trommel for further grading. Thus, several trommels may operate in series. At one of the newer mills the trommel is equipped with 2 sizes of screen cloth, which make 1 oversize and 2 undersize grades. Any number of grades may be made, and special grades may be obtained by blending. Sand and dust are removed from the graded fiber on flat cleaning screens. Finished fiber lifted from the cleaning screen by air suction is conveyed to the store room for bagging.

Classification of Fiber

Methods of grading asbestos fiber by length have been described in a preceding section. To determine with reasonable accuracy the quality of fiber in each grade and thus be able to modify grading methods accordingly, a standard testing machine is used. The purchaser can rely on the standard test and thus avoid any confusion that might ensue from dependance upon trade names or designations. A description of the Standard testing machine[6] follows:

> The machine consists of a nest of 4 wooden boxes, measuring 24-1/2 by 14-3/4 inches and 3-1/2 inches in depth. The boxes, which are superposed one above the other, are numbered from the top down 1, 2, 3, and 4. The bottoms of boxes nos. 1, 2, and 3 are made of metallic screen of the following specifications: Box no. 1: 1/2-inch opening, diameter of wire, 0.105 inch. Box no. 2: 4-mesh wire, 0.063 inch. Box no. 3: 10-mesh wire, 0.047 inch. Box no. 4 is a receptacle for the fines which fall through the three other boxes. The nest of 4 boxes or trays rests on a table to which an eccentric with a throw of 25/32 inch gives a movement of 1-9/16-inch travel.
>
> To made a test, 16 ounces of asbestos is put on the top tray which is covered. The machine is run at the rate of 300 r.p.m. at the shaft of the eccentric, and by means of an automatic device this is kept going for exactly 2 minutes, giving the nest a horizontal shaking movement. At the end of this time the asbestos which remains on each tray is weighed. This gives the grades of the asbestos fiber; the longest fiber naturally stays on the top tray, whereas the shorter fiber, according to its length, remains on screens 2 and 3 or drops into the pan or lowest tray. The more fiber retained on the first screen and the less fiber in the pan, the higher the grade and therefore the greater its value. If, for instance, a customer buys spinning fiber of the specification 4-7-4-1, it means that in a sample of 16 ounces, representing the average of the lot shipped, 4 ounces will remain on the top screen, 7 on the second, 4 on the third and finally 1 ounce will go through all the screens into the pan. He will evidently pay more for this material than for paper stock testing 0-0-10-6. This indicates that out of 16 ounces tested nothing is retained on the first 2 screens, 10 ounces remain on the third, and 6 ounces go through the latter into the pan. It is evident that the figures of the test represent the proportion in ounces of the different lengths of fiber in a pound of asbestos.

Samples for testing usually are taken at the grading machines every half hour. Owing to unavoidable variations in fiber as it comes from the pit, the quality of mill-run fiber usually is maintained a little higher than its designation.

6/ Ross, J. G., work cited, p. 49.

For many years each Canadian company graded its fiber according to its own standards and sold the products under its own trade designations. This practice led to much confusion in marketing, because different mills employed similar marks for grades quite unequal in quality and value. It was frequently necessary for buyers to have samples tested before placing orders. In 1931 producers in Quebec agreed upon a uniform classification of fibers, whereby they are divided into nine groups, and each group divided into grades. Crude asbestos was defined as "hand-selected cross-vein material essentially in its native or unfiberized form," and milled asbestos as "all grades produced by mechanical treatment of asbestos ore." Except for the very lowest grades, which are based on the weight per cubic foot, all milled grades are based on the results of tests in the standard testing machine previously described. Ross[7] gives a complete description of the standard grades of Canadian asbestos as agreed upon in 1931:

Crude asbestos

Class	Standard designation of grade	Description
Group No.1	Crude No.1	Consists basically of crude 3/4-inch staple and longer.
Group No.2	Crude No.2	Consists basically of crude 3/8-inch staple up to 3/4-inch.
	Crude run-of-mine	Consists basically of unsorted crudes.
	Crudes sundry ...	Consists of crudes other than above specified.

Milled asbestos

Group No.3	Spinning or textile fiber ...	Consists of fiber testing 0-8-6-2 and over.
Group No.4	Shingle fiber ..	Consists of fiber testing below 0-8-6-2 to and including 0-1½-9½-5.
Group No.5	Paper fiber	Consists of fiber testing below 0-1½-9½-5 to and including 0-0-8-8.
Group No.6	Waste, stucco or plaster.......	Consists of material testing below 0-0-8-8 to and including 0-0-5-11.
Group No.7	Refuse or shorts	Consists of material testing 0-0-5-11 and below, including material testing below 0-0-1-15 and specified as weighing 35 pounds or less per cubic foot, loose measure.
Group No.8	Sand	Consists of such asbestos mill products as sand weighing over 35 pounds per cubic foot, loose measure, and under 75 pounds per cubic foot, loose measure and containing a preponderance of rock.
Group No.9	Gravel and stone	Consists of such asbestos mill products weighing 75 pounds and over per cubic foot, loose measure.

(Continued on next page.)

7/ Ross, J. G., work cited, pp. 50-51.

Subdivision of the groups of milled asbestos

Group no.	Standard designation of grades	Guaranteed minimum shipping test
No. 3: Textile and spinning fibers	3D	8 - 6 - 1 - 1
	3F	7 - 7 - 1½ - ½
	3K	4 - 7 - 4 - 1
	3M	2 - 9 - 4 - 1
	3R	2 - 8 - 4 - 2
	3T	1 - 9 - 4 - 2
	3Z	0 - 8 - 6 - 2
No. 4: Shingle fibers.............	4D	0 - 5 - 10 - 1
	4F	0 - 3 - 12 - 1
	4K	0 - 4 - 9 - 3
	4M	0 - 4 - 8 - 4
	4R	0 - 3 - 9 - 4
	4T	0 - 2 - 10 - 4
	4Z	0 - 1½ - 9½ - 5
No. 5: Paper fiber................	5D	0 - ½ - 10½ - 5
	5F	0 - ½ - 9½ - 6
	5K	0 - 0 - 12 - 4
	5M	0 - 0 - 11 - 5
	5R	0 - 0 - 10 - 6
	5T	0 - 0 - 9 - 7
	5Z	0 - 0 - 8 - 8
No. 6: Waste, stucco, or plaster..	6D	0 - 0 - 7 - 9
	6F	0 - 0 - 6½ - 9½
No. 7: Refuse or shorts...........	7D	0 - 0 - 5 - 11
	7F	0 - 0 - 4 - 12
	7H	0 - 0 - 3 - 13
	7K	0 - 0 - 2 - 14
	7M	0 - 0 - 1 - 15
	7-20	20 lb.per cu. ft.,loose measure.
	7-25	25 lb.per cu. ft.,loose measure.
	7-30	30 lb.per cu. ft.,loose measure.
	7-35	35 lb.per cu. ft.,loose measure.
No. 8: Sand......................	8-40	40 lb.per cu. ft.,loose measure.
	8-45	45 lb.per cu. ft.,loose measure.
	8-55	55 lb.per cu. ft.,loose measure.
	8-75	75 lb.per cu. ft.,loose measure.
No. 9: Gravel and stone..........	9	75 lb. and over per cubic foot, loose measure.

The fiber is packed in jute bags of 100 or 125 pounds capacity marked with the grade letter or number and the manufacturer's name.

Arizona

The high cost of transportation to railways and heavy freight charges to markets render the sale of shorter-grade Arizona fibers unprofitable. Spinning fibers are therefore the principal products of the Arizona deposits. The smaller mines produce hand-cobbed crudes almost exclusively. The rock is mined and cobbed by the same workers on a contract basis. Cobbing is conducted in much the same way as in Canada.

Three of the larger mines have milling facilities. The largest mill, equipped with crushers, cyclones, and screens, produces No. 1. mill fiber 3/4 to 2-1/2 inches long; No. 2, 3/8 to 3/4 inch long; No. 3, 1/8 to 3/8 inch long; and No. 4, 1/8 inch and less. The products are graded according to tests made with a standard Canadian testing machine.

In one of the smaller mills the rock first passes through a jaw crusher followed by two sets of rolls. Two grades of fiber are separated by an impact screen. The shorter asbestos is treated with a small fiberizing machine and is cleaned and graded in a trommel 16 feet long and 4 feet in diameter.

Another small mill produces 5 grades of spinning fibers and a shingle stock. No grades shorter than shingle stock can be sold profitably.

A considerable quantity of rock containing spinning-fiber veins, which are difficult to cob by hand, are treated in these mills and yield what is known as mechanically cobbed crude fiber, an anomalous designation according to the Canadian definition of crude, which includes only hand-cobbed fiber. The claim is made, however, that the crude fiber can be graded by machine more accurately than by hand.

Vermont

The asbestos-bearing rock quarried near Eden, Vt., first is crushed to 5- or 6-inch size in a 24-by 36-inch jaw crusher and then discharged to a 36-inch belt conveyor, which carries the material to a 1,000-ton storage bin. When taken from storage it is crushed to 3- or 4-inch size in a 13-by 30-inch jaw crusher, passed through a rotary drier, and again crushed to 1- or $1\frac{1}{2}$-inch size in a gyratory crusher, whence it is conveyed to a second storage bin. The product of the tertiary crusher is treated in a series of Jumbos or fiberizers like those used in Quebec, and the fiber is recovered by screening and air suction. After passing through dust removers and graders it falls into bins according to its length and is bagged in 100-pound burlap bags. Three principal products are manufactured: (1) Shingle stock, testing 0-2-10-4 or 0-$1\frac{1}{2}$-$9\frac{1}{2}$-5; (2) millboard, paper or molded brake-lining fiber testing 0-0-10-6; and (3) material suitable for roofing, paints, plastics, molded products, and boiler covering, testing 0-0-5-11. Occasionally a higher-grade fiber, testing

0-8-6-2, suitable for magnesia pipe covering is produced. Virtually no crude or spinning fibers are produced in Vermont.

Union of South Africa

Chrysotile. - At the New Amianthus mine, Barberton district, Eastern Transvaal, long-fibered chrysotile is hand-sorted and graded by natives as A and B, fiber over 1-1/8 inches long and fiber 3/4 to 1-1/8 inches long, respectively; both are designated as crudes. A third grade, E, consisting of hand-selected fiber down to a half inch long might be classed as crude.

Rock bearing the shorter fibers is crushed, dried, and reduced by rolls. The product of each set of rolls is sieved over 10-mesh screens, the fines going to waste. After passing through four sets of rolls the product is virtually pure fiber except for dust adhering to it. The asbestos is passed through a fiberizer to beat out the dust, then over a dust sieve, and finally over grading sieves. Two principal mill grades are made: F, 1/4 to 1/2 inch; and G, 1/5 to 1/4 inch. Grade F corresponds to Canadian fiber testing 0-4½-10-1½. The milled fibers are principally shingle and paper stocks. The mill has a capacity of 6,000 tons of rock a month.

At the Munnik-Myburgh mine in the same district 4 grades of hand-cobbed chrysotile fiber are designated as follows: IXL, over 3/4 inch; XL, 1/2 to 3/4 inch; X, 1/4 to 1/2 inch; and XX, 1/8 to 1/4 inch. Hand-cobbed fiber is graded in shaking screens.

Short fiber is treated in a mill with an elaborate flow sheet. Dried and crushed rock is passed through a series of rolls followed by grading screens. All five products obtained are passed through a regrading plant. The grades and their values, as shown on a Canadian standard testing machine, are as follows:

M1	2-11-2-1
M2	0-8½-6-1½
M3	0-3-10-3
M4	0-0-8½-7½
M5	0-0-4½-11½

The milled grades constitute about 95½ percent of the total fiber recovery.

Crocidolite. - Methods of milling and grading crocidolite vary greatly in different localities. In the Southern Belt of the Cape Provinces the workings are numerous, and many are relatively small. At the smaller workings the fiber is all hand-cobbed. It is often hand-sorted into grades according to length, but at the larger workings the hand-cobbed asbestos is graded by machinery. A hand-turned trommel made of perforated sheet iron or covered with wire netting usually is employed. The cobbed fiber from a series of workings may be shipped to a central screening plant where all asbestos 1/2 inch long or over is recovered, and the minus 1/2-inch product is taken to a mill for further crushing and grading. At one mill the rolls travel at slightly different speeds so as to subject the fiber to a tearing action. Double grading trommels are sometimes used. The inner cylinder has 1/4-inch holes

and the outer 1/8-inch. Material passing the outer cylinder is waste, the oversize from the outer cylinder is designated grade SS, and the oversize from the inner cylinder grade S. The following grades are produced:

E: Over 2-inch cobbed fiber.
D: 1-3/4-inch to 2-inch cobbed fiber.
C: 1-1/4-inch to 1-3/4-inch cobbed fiber.
B: 7/8-inch to 1-1/4-inch cobbed fiber.
A: 1/2-inch to 3/4-inch cobbed fiber.
S: 1/4-inch to 1/2-inch milled fiber.
SS: Shorts, 1/8-inch to 1/4-inch milled fiber.

In the Northern Belt material over three fourths inch long is hand-cobbed. For many years the mill grades were not standardized, and the companies had difficulty in marketing their products. Later a large mill was erected at Kuruman, and the grades manufactured there established standards that offered such decided marketing advantages that other producing companies adopted them. In this mill rock carrying the shorter fibers is passed through crushers, heavy rolls, and disintegrators. The fiber thus separated is classified on grading screens. The following grades are made:

ES: Over 1-1/4-inch hand-cobbed fiber.
No. 1: 3/4-inch to 1-1/4-inch hand-cobbed fiber.
Also 2 grades of plus 3/4-inch discolored fiber -
No. 2: 3/8-inch to 3/4-inch milled fiber.
No. 3: 3/8-inch, or under, milled fiber.

Of the total fiber recovered about 13-1/2 percent is hand-cobbed and 86-1/2 percent milled. In both the Northern and Southern Belts about 9 percent of the total fiber is of spinning grade.

Crocidolite of the Pietersburg district, Transvaal, is passed through rolls, fiberizers, and grading trommels. Four grades are produced:

B: Over 1-1/4-inch.
A: 5/8-inch to 1-1/4-inch.
S: 1/4-inch to 5/8-inch.
Paper grade: 1/8-inch to 1/4-inch.

Amosite. - Because of its unusual length most of the amosite produced is hand-cobbed. The fiber thus freed from adhering rock is passed through a series of rolls and disintegrators and then graded on shaking screens or trommels usually into three grades. No. 1 is spinning fiber; No. 2 is used in the manufacture of rope, felts, and similar products; and No. 3 is used for asbestos-cement products, such as roofing slates and corrugated sheathing. The fiber is classified both according to length and color. The colors range from whitish gray to yellowish; the gray shades are considered the best. Most of the mills are small.

Rhodesia

The Rhodesian asbestos industry is centered at the large chrysotile mines of Shabani. To avoid damage to the fiber as much of it as possible is separated from the rock close to the working face; hence the hand-cobbing and bagging of crude fiber is an important part of quarrying and mining operations. Each cobbing boy recovers about 1,500 pounds of fiber a shift in open quarries and 1,000 pounds a shift in stopes. The cobs are said to average 40 to 50 percent fiber.

The quarry rock first is broken in jaw crushers and rolls, and the products are sorted on picking belts into three groups: (1) Rock containing little or no fiber, (2) rock containing large seams of fiber, and (3) fines and rock carrying small seams of fiber. Product 1 is left on the belt and conveyed to the waste heaps, product 2 is carried on a second belt to the cobbing sheds, and product 3 is conveyed by a third belt to the mill. Hand cobs from both the cobbing sheds and the quarries or stopes are fiberized and graded in one mill, and the fiber-bearing rock is reduced in another.

Some maintain that when fiber-bearing rock is kept wet until it reaches the mill it suffers less damage and is in better condition for milling than when it is handled and rehandled dry. After primary crushing in the mill the rock is dried either with steam-heated equipment or in rotary driers in which the products of combustion from wood fires are in direct contact with the rock. The dried rock is reduced in a series of small grinding pans operating like pug mills. Fiber is separated from rock with shaking screens and air suction in the same manner as in Canada. Undesirable brittle fiber is ground in the pans and passes through the screens with the waste rock. Fiber carried in the air current is deposited in a centrifugal settler, and the air passes on through the exhaust fan and dust trap to an outlet. The asbestos is taken to a grading mill, where it is cleaned further from dust and separated into the various market grades. Grading is done in hexagonal trommels.

Cobbed asbestos is fiberized, cleaned, and graded in a mill in which the equipment is modified to handle longer fiber.

In the Shabani area the tonnage of cobbed fiber ranges from 2-1/2 to 3-1/4 percent of the total rock mined. Total milled and cobbed fiber averages about 4 percent of the rock quarried. One must not infer that the rock is leaner than in Quebec, whererecovery is about 6 percent, because short-fiber material amounting to many thousand tons in Quebec is discarded as waste in Rhodesia.

Few data are available on the grades produced at Shabani or on the proportion of each grade. From 25 to 30 percent is spinning fiber, and most of the remainder is shingle stock. Fiber shorter than paper stock cannot be handled profitably because of the heavy transportation expense to markets. The longest spinning fibers, Nos. 1 and 2, are the only grades quoted in the United States market reports.

At the King's and Gaths' mines in the Mashaba district the asbestos is recovered by hand-cobbing only. The tonnage of marketable fiber obtained is 0.7 to 1.2 percent of the rock quarried.

Russia

In Russia crude asbestos is sorted in the quarry and cobbed as in Canada, but with increasing mechanization hand-cobbing has declined, and a larger proportion of the fiber is prepared mechanically. For many years the fiber was concentrated by hand-sorting in the quarries. Although recoverable fiber averages about 4-1/2 percent of total rock quarried the elimination of barren rock by hand-sorting so raised the grade that mill feed averaged about 22 percent of the fiber. During recent years mechanical mining equipment has replaced hand methods to a large extent, and the fiber is concentrated by hand-sorting on picking belts after the rock is crushed.

The preliminary stages of rock treatment are exemplified best in one of the newer mills, the "Gigant", which began operation in 1932. Rock from the quarry is conveyed to a storage bin and from that to a No. 9 Gates gyratory crusher, which reduces it to about 6-inch size. It discharges to a picking belt, where about 20 percent of the primary feed containing too little fiber to justify milling is thrown out as waste rock. The asbestos-bearing rock is reduced to 1-1/2- to 2-inch size in two No. 7 Gates gyratory crushers discharging to heavy shaking screens. The oversize from the screens passes to picking belts, where more barren rock is eliminated. The good rock from the picking belt is reduced to about 1-inch size in a jaw crusher and joins the undersize from the shaking screens in a wet-storage bin whence it passes to 3 rotary driers, which reduce the water content from about 8 to 1 or 2 percent. The product is conveyed to the dry-storage bin.

The most noteworthy feature of the preliminary milling stage is the concentration on picking belts. Because the serpentine tends to break cleanly along the fiber veins, effecting a more or less distinct separation of barren and fiber-bearing rock, conditions particularly favor this method of concentration. Of the original mill feed, consisting of 2,400 tons a day, about 1,440 tons are eliminated as waste, leaving only 960 tons for the later milling processes. Thus, picking belts save operators a great deal of useless milling.

In the more advanced milling stages the dried rock is passed over heavy shaking screens, from which four products are obtained: (1) Fiber removed by suction fans, (2) oversize rock conveyed to a set of rolls, (3) middlings that bypass the rolls and are carried to the next screen, and (4) fines conveyed to a Humboldt disintegrator. By means of a series of such rolls, screens, suction pipes, and disintegrators virtually all of the fiber is recovered. Fiber from the collecting hoppers is sent to a series of shaking screens for cleaning. The cleaned fiber is collected again by suction fans and classified by length in slowly rotating grading trommels.

The older mills employed a typically Russian method of milling. The rock was reduced in a series of rolls, and part of the fiber was separated from rock with trommels; the major separation, however, was made on inclined planes provided with suitable gaps. The fiber dropped through the gaps, and the rock fragments jumped over them. Ru Keyser[8] has described milling processes in some detail.

A large new mill designed to handle 2,000,000 tons of rock annually and produce 80,000 tons of fiber in 6 grades was nearing completion in 1934. This mill will give Russia a total milling capacity of approximately 175,000 tons of fiber a year.

Little information is available regarding the grades produced and the percentage of each grade. Estimates made in 1931 indicated that fiber one third inch or more in length constituted about 30 percent of the total output. Since that date, with increasing factory capacity for the manufacture of asbestos products, probably a larger proportion of the shorter grades is produced. Grades entering the German market in 1929 were designated as follows:

Grade 0: 1-1/2 inch or longer.
Grade 1: 3/4 inch to 1-1/2 inches.
Grade 2: 5/8 to 3/4 inch.
Grade 3: 3/8 to 5/8 inch.
Grade 4: 1/8 to 3/8 inch.
Grade 5: Less than 1/8 inch.

Price quotations of Russian fiber in the American market include Crude No. 1, Crude No. 2, Crude No. 3, and shingle stock.

Cyprus

In Cyprus the rock-bearing chrysotile asbestos is so decomposed that it breaks down readily into small sizes. Small quantities of hand-cobbed fiber are obtained. Rock reduced in the quarry by hand methods is screened to eliminate barren material over 18 mm (7/10 inch) and fines under 5 mm (1/5 inch); the intermediate product, which averages about one sixth of the total quarry output, is sent to the mills. The mill feed is crushed and passed over flat shaking screens and the fiber recovered by air suction. It is treated further to remove dust and to classify it according to length. Recovery of fiber is about 3-1/2 percent of the mill feed, and virtually the entire production is short fiber.

Milled asbestos is graded into three classes - standard, shorts, and fines. The standard grade, designated "shingle stock", is said to include about 75 percent of the production. The entire output is exported. Twelve mills were reported in operation in 1935. Their aggregate capacity is 25,000 tons of fiber a year of 6 working months. They are operated during the dry season only.

8/ Ru Keyser, Walter A., Asbestos Milling in the Urals: Eng. and Min. Jour., vol. 134, no. 10, October 1933, pp. 415-419.

Australia

The Lionel plant in Australia produces four grades, which are numbered in the reverse order from that generally practiced; the smaller numbers are applied to the shorter fibers. No. 1, the lowest grade produced, includes fibers about 1/4 inch long; No. 2, about 1/2 inch; No. 3, about 1 inch; and No. 4, 2 inches and over.

Anthophyllite Milling

Anthophyllite, a variety of amphibole asbestos, has been mined in a small way in the United States for many years. Small mills have been operated in Georgia, Montana, North Carolina, Oregon, Virginia, and Washington. As a large percentage of the rock treated is asbestos, fiberization and classification are the principal milling processes. The fiber is much weaker than chrysotile, and therefore severe treatment is to be avoided.

In one mill operated a few years ago the dried rock was broken in a gyratory crusher, whence it went to a hammer mill equipped with air suction for carrying off the loose fiber. This process was followed by reduction in a double-cage mill, the cages running in opposite directions. The fiber was removed by air suction; an enlargement in the suction pipe retarded the velocity of the air current, permitting heavy fragments of unopened fiber to fall. These fragments were returned through a trap to the cage mill in closed circuit. The fiber was classified with vibrating screens.

In another mill the dried rock is first reduced with a jaw crusher, followed by a set of rolls. The crushed asbestos rock is fiberized in a hammer mill equipped with an air separator. Coarse material is returned to the fiberizer, dust is collected in a dust chamber, and the fiber is conveyed to a bin that feeds into a packer. The fiber is packed in 100- or 200-pound sacks. Modifications of these processes are employed in other mills.

MARKETING

Asbestos is used in so many diverse products that its markets are numerous and widespread. It is used extensively in the United States, England, France, Germany, Belgium, Italy, Netherlands, Spain, Russia, and Japan. The last country has become an important buyer of roofing-grade asbestos. Russia at one time exported almost its entire production, but it recently has become an important manufacturer of asbestos products. Many other countries use smaller quantities. With greater diversity in use, expansion of automobile production, development of new types of building materials, and stronger demands for heat-insulation material the consumption of asbestos has increased steadily. In recent years mineral, slag, and glass wools have increased in importance as insulation, and although generally used in fields other than those in which asbestos is employed they are replacing asbestos to some extent.

Canadian asbestos generally is preferred by users in the United States, although African and Russian fibers may be substituted for many uses. Considerable quantities of the latter fibers are mixed with the Quebec product.

The principal Canadian producers sell direct to consuming industries and also to dealers and agents who distribute it to consumers. Sales frequently are made to consumers only after they have examined and approved samples submitted by dealers or producers. Canadian firms sometimes supply European customers with trial consignments of several tons, particularly if they are introducing new products. Sales are also made according to fiber length as tested and guaranteed by the producer. The larger manufacturers of asbestos products have their own testing machines and can check the producer's classification. The Canadian standard testing machine is used to some extent in Africa.

Asbestos is sold in 100-pound or 125-pound bags on a short-ton basis, bags included. Quotations are f.o.b. mines. The weight of a given volume varies with the length of fiber as the longer fibers are bulkier. The volume of a short ton ranges from 60 to 90 cubic feet. A minimum carload of fiber is 20 tons and of asbestic, 30 tons. Settlements usually are made on the basis of 2 percent discount within 10 days or 30 days net.

Before the World War Hamburg was the principal marketing center, but since that time New York has become the most important. The larger producers now have offices in New York, Hamburg, London, and other large cities.

Market requirements are based principally on fiber length, but strength, flexibility, color, chemical composition, and cleanliness may have an important bearing on use. The principal market outlets are indicated in the following brief summary of uses.

The longer and more valuable asbestos crudes and fibers, ranging upward from about $70 a ton, are used in manufacturing woven brake linings, textile fabrics, and asbestos packings. The next lower grades, ranging from about $25 to $70 a ton, are used in manufacturing rigid asbestos-cement shingles, lumber, and corrugated sheathing. Still shorter fibers are used in manufacturing paper and millboard, and the lowest-grade fibers are employed in heat-insulating cements, cold molded products, and fillers.

European imports from Africa are handled principally by dealers or agents. The importers pay cash or open a credit in South Africa and sell on credit to manufacturers. The latter concerns do not buy directly from producers because they are accustomed to buy on 3 to 6 months' credit, and importations are not made easily on such a basis.

Several of the largest asbestos organizations are of the vertical type; that is, they mine and mill the raw materials and fabricate finished products. At least three large companies in the United States are of this type, each having mines in Canada and factories in the United States. Turner & Newall, Ltd., controls nearly all of the asbestos production of Rhodesia and much of

that of the Union of South Africa and has 20 large asbestos-products factories in England. The Russian industry is also of this type; the State controls both production of raw materials and manufacture of finished products. Because of this condition a substantial portion of the asbestos consumed never enters the open market.

Most of the world supply of raw asbestos is in exceptionally strong hands, and distribution is conducted in close and sympathetic collaboration with buyers, many of whom are also strong, well-organized concerns. Hence, new producers and new sales organizations are not likely to dislocate the market seriously. Furthermore, new organizations sometimes have difficulty in finding a market because consumers of asbestos ordinarily place orders 12 to 18 months in advance of actual needs. Centralized control by large units also makes possible the establishment of marketing agreements. Turner & Newall, Ltd., having control of virtually the entire output of Rhodesia, was able in 1931 to consummate an agreement with Russia whereby orders were apportioned between European consumers and the market stabilized on a basis satisfactory to all concerned.

PRICES

Prices of Canadian asbestos have fluctuated greatly during the past 15 years. In 1920, because of war stimulation, prices skyrocketed to unprecedented heights, Crude No. 1 selling for more than $3,000 a ton. In 1921 the price dropped to less than half that amount, and by 1925 the highest grades were selling for only about one eighth of the price received in 1920. Table 1 shows, except for 1933 and 1934, the average price per short ton f.o.b. mine received by producers as reported to the Quebec Bureau of Mines. These figures generally are considerably lower than the average price quotations as given in Mineral and Metal Markets. Since 1932 the Canadian figures, unfortunately, are not presented in enough detail to show values by grades.

Price trends are shown graphically in figures 1 and 2.

Average quotations for Vermont asbestos, as given in Mineral and Metal Markets, are shown in table 2. Prices are per short ton f.o.b. mines.

Table 3 shows prices of Rhodesian and Russian asbestos per short ton, average quotations for the year, as shown in Mineral and Metal Markets. All prices are c.i.f. New York.

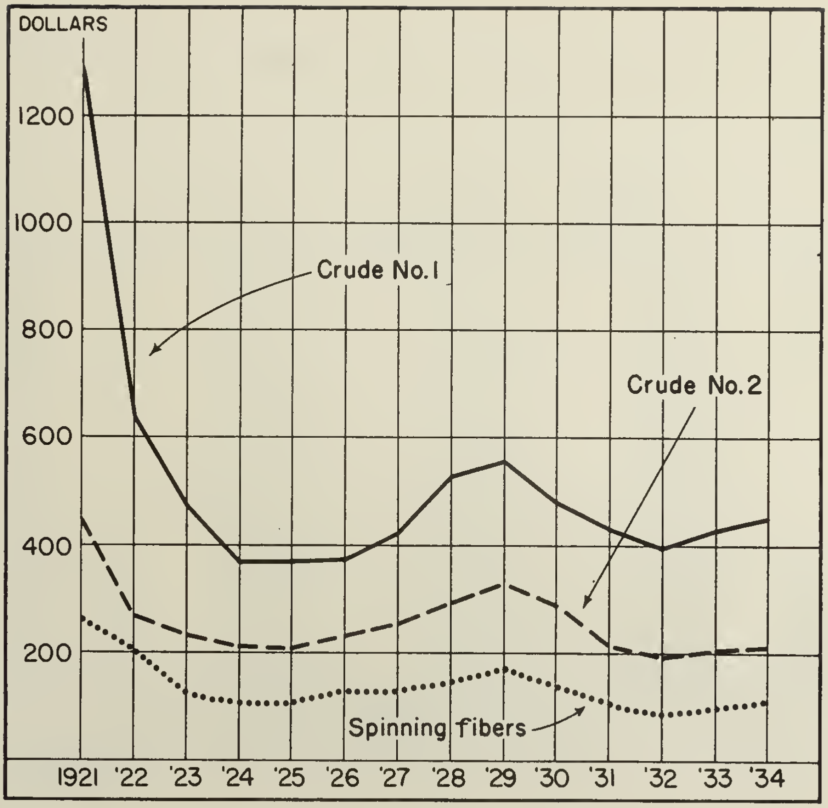

Figure 1.- Prices of Canadian crudes and spinning fibers, 1921-34. Data for 1921-32 from value of sales compiled by the Bureau of Mines, Province of Quebec. Data for 1933 and 1934 from price quotations in Mineral and Metal Markets.

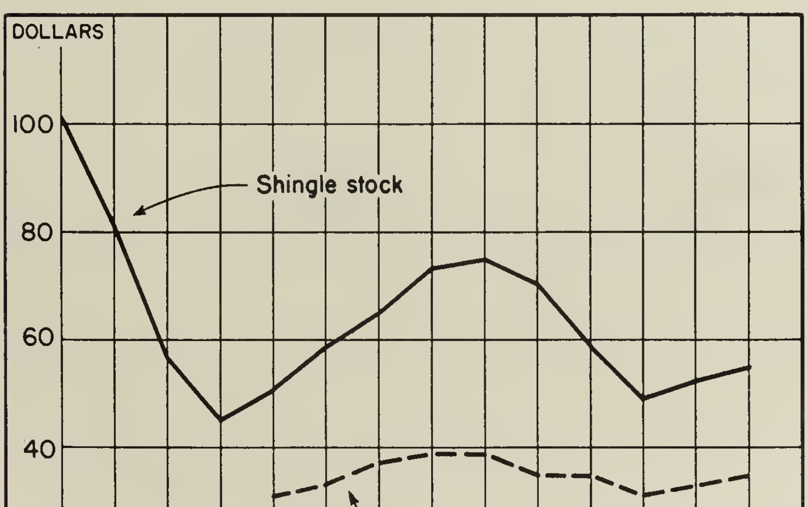

Figure 2.- Prices of Canadian shingle, millboard, and paper stocks, 1921-34. Data for 1921-32 from value of sales compiled by the Bureau of Mines, Province of Quebec. Data for 1933 and 1934 from price quotations in Mineral and Metal Markets. Figures for millboard and paper not available prior to 1925.

TABLE 1. - Average value per short ton of asbestos sold in Canada, 1921-34 1/

Year	Crude No. 1	Crude No. 2	Spinning Fibers	Shingle Stock	Millboard and paper stock
1921	$1,281.32	$ 446.91	$ 263.09	$ 101.75	2/ $ 31.19
1922	648.68	265.32	207.71	81.00	2/ 21.65
1923	472.60	237.29	123.37	57.05	2/ 22.01
1924	365.97	215.27	110.81	45.12	2/ 19.84
1925	364.96	206.12	106.43	50.78	31.03
1926	371.51	229.62	130.22	58.62	33.88
1927	423.65	249.59	129.32	64.80	37.82
1928	534.87	296.65	148.71	73.80	38.73
1929	557.38	331.82	177.30	75.26	38.56
1930	480.44	285.44	141.52	70.64	35.33
1931	431.46	216.35	107.22	58.72	3/ 34.66
1932	396.94	192.43	91.36	49.64	3/ 30.84
1933	437.50	206.25	100.20	52.71	3/ 32.71
1934	450.00	212.50	112.50	55.00	3/ 35.00

1/ Figures for 1921-32 from Quebec Bureau of Mines; for 1933-34, average of quotations, Mineral and Metal Markets.
2/ Paper and others.
3/ Paper only.

TABLE 2. - Prices of Vermont asbestos, 1931-34

Year	Shingle stock	Paper stock	Cement stock
1931	$ 50.71	$ 35.00	$ 20.00
1932	42.92	32.92	20.00
1933	42.92	32.71	20.25
1934	45.00	35.00	23.00

TABLE 3. - Average price quotations per ton of foreign asbestos

Year	Rhodesian		Russian			
	No. 1	No. 2	Crude No. 1	Crude No. 2	Crude No. 3	Shingle stock
1923	$361.11	$227.77				
1924	313.75	193.75				
1925	292.50	222.91				
1926	325.00	250.00				
1927	402.08	322.91				
1928	450.00	350.00				
1929	450.00	350.00				
1930	383.33	300.00	$210.00	$160.00	$ 50.00	
1931	304.17	204.17	227.50	184.17	113.54	
1932	187.50	131.25	213.54	163.54	125.00	
1933	186.67	136.67	212.50	162.50	125.00	$ 55.00
1934	210.00	160.00		162.50	128.75	$ 55.00

Asbestos-Products Plants

United States. - The United States leads all countries in the manufacture of products of which asbestos is an essential or important constituent. The Bureau of the Census listed 238 establishments in the United States making asbestos products in 1929, 208 in 1931, and 187 in 1933. The industry is centered chiefly in the North Atlantic States, notably in the Philadelphia (Pa.) and eastern New Jersey areas. Important factories are operated also in Cincinnati (Ohio), Chicago (Ill.), St. Louis (Mo.), and other midwestern cities; in the Southern States; and on the Pacific coast. Most of the output enters the domestic market; less than 5 percent in value is exported.

Table 4 shows the quantity and value of the principal asbestos products manufactured in the United States in recent years, as given in the Biennial Census of Manufacturers. The articles listed as "other products" are very numerous; some manufacturing companies sell more than 200 different articles. Many products included under the last four items in the table consist of various materials, including wood, steel, copper, cork, and rubber, in addition to asbestos, which may form a very small proportion of the total.

TABLE 4.- Asbestos products manufactured in the United States, 1929-33
(Data from Bureau of Census)

	1929		1931		1933	
	Quantity	Value	Quantity	Value	Quantity	Value
Asbestos textiles and textile products:						
Brake linings.......................feet	110,246,479	$16,447,438	56,059,659	$ 8,367,492	35,472,491	$ 4,545,006
Clutch facings.....................pieces	38,096,113	3,910,727	21,340,773	1,796,761	18,857,760	1,601,934
Yarn..............................pounds	15,378,407	5,173,568	9,512,686	2,471,388	8,487,342	1,551,909
Cloth............................ do.	6,666,589	2,767,049	4,629,158	1,413,959	4,648,677	1,069,751
Tape, listings and tubular lagging do.	2,225,852	1,202,231	2,350,548	880,538	1,735,925	580,948
Packing.......................... do.	8,623,281	3,845,710	6,747,122	2,333,956	4,971,327	1,883,940
Gaskets.......................... do.	3,012,335	1,726,804	1,441,325	644,385	1,400,192	577,177
Other textiles............................	-	1,088,466	-	630,770	-	812,535
Total textiles and products............	-	$36,161,993	-	$18,539,249	-	$12,623,200
Asbestos building materials:						
Shingles.........................squares	894,164	$ 5,277,308	565,997	$ 3,266,054	408,256	$ 1,826,279
Lumber......................square feet	35,395,194	3,094,426	16,907,820	1,563,135	14,193,900	1,032,313
Other building materials................	-	2,051,884	-	818,343	-	620,546
Total building materials...............	-	$10,433,618	-	$ 5,647,532	-	$ 3,479,138
Other asbestos products:						
Brake lining, molded.................feet	22,410,872	$ 2,534,646	24,180,184	$ 2,823,738	24,373,952	$ 3,422,593
Brake linings (kind unrecorded)..........	-	-	-	3,116,045	-	2,574,441
Pipe and boiler covering, air-cell...feet	67,331,323	4,657,666	39,083,826	2,755,354	19,554,455	1,137,283
Pipe and boiler covering other than air-cell.......................... do.	27,686,503	2,516,009	8,797,834	938,149	8,460,631	851,826
Pipe and boiler covering, 85 percent magnesia........................... do.	23,959,209	3,045,324	19,323,303	2,511,795	10,818,354	1,457,211
Molded blocks, 85 percent magnesia.Bd.ft.	21,479,103	2,457,357	11,740,953	1,209,015	9,711,170	852,844
Millboard.........................pounds	7,280,221	345,375	-	-	-	-
Insulating cements................ do.	71,197,630	1,017,964	39,100,027	602,853	34,456,331	625,999
Table mats and protectors......... do.	1,615,631	766,222	-	-	-	-
Other products 1/.........................	-	5,105,960	-	3,485,101	-	1,578,547
Gaskets other than asbestos textile 2/...	-	15,741,964	-	10,153,121	-	8,935,839
Metallic and semimetallic packings.......	-	-	-	2,110,725	-	1,809,361
Steam and other packing other than asbestos textile 3/....................	-	10,999,336	-	2,272,013	-	2,250,584
Total other products................	-	$49,187,803	-	$31,977,909	-	$25,496,528
Grand Total.........................	-	95,773,414	-	56,164,690	-	41,598,866

1/ Asbestos paper and miscellaneous products; includes millboard in 1931 and 1933.
2/ Includes some gaskets that contain little or no asbestos.
3/ Excludes leather packings.

Canada. - For many years Canada exported almost its entire production of raw asbestos but lately has developed an important asbestos-products industry. In 1933, 11 plants were engaged in this line of manufacture. Four of them, in the center of the asbestos-mining district in Quebec, accounted for 77 percent of the total production. There are also 5 plants in Ontario and 1 each in Nova Scotia and British Columbia. The capital of these concerns was estimated at more than $1,777,000, and the selling value of the products at the works in 1933 was $757,626. The principal products are brake-band linings, boiler and pipe coverings, packings. and paper and roofing.

Other countries. - Many of the data presented herein, except those relating to Great Britain, were assembled by Bruckner in 1929.[9]

The manufacture of asbestos products is an important industry in Great Britain. In 1929, 57 factories were operated by 35 companies, and the estimated value of their output was $28,163,000. Of the raw asbestos used 44 percent came from Rhodesia, 28-1/2 percent from the Union of South Africa, 21-1/2 percent from Canada, and small amounts from Cyprus, the United States, Italy, and Russia. The range of products is much the same as in the United States. About one fourth of the total output is exported.

In 1929, 32 factories in Germany made asbestos products, 17 of which were devoted chiefly to spinning, weaving, and braiding. Four factories made corrugated sheeting, pipe, roofing tile, and other asbestos-cement products. The raw asbestos is obtained chiefly from Russia.

In Austria, the original home of "Eternit", 9 asbestos-products factories were operated in 1929; asbestos-cement tile was the principal product of 4 factories.

In the same year 4 factories in France produced asbestos-cement roofing tiles and similar products; 9 plants were devoted chiefly to the manufacture of spun, woven, and braided products; 4 factories made high-pressure packings; and 2 factories made asbestos brake linings.

Nine factories in Belgium in 1929 were devoted principally to the manufacture of asbestos-cement goods, with a minor output of technical articles. In Italy 3 large factories and numerous small ones made chiefly asbestos-cement tiles and sheets. Thread, cards, cartons, textiles, and filter paper were also made in considerable quantities. Three factories in Czechoslovakia were devoted to spinning, weaving, and braiding; 1 made asbestos board; 4, high-pressure packings; and several, asbestos-cement tile. Most of the plants are small.

One factory established at Helsingfors, Finland, in 1923 manufactures "Eternit" products, packings, insulating materials, and fabrics. Both domestic and foreign asbestos is used.

9/ Bruckner - The Continental Asbestos Industry: India-Rubber Jour., international issue, Oct. 31, 1929, pp. 19-21.

According to recent data about two thirds of the Russian output of asbestos is used in domestic factories devoted to the manufacture of asbestos products. Russia therefore is becoming an important factor in this industry. One very large factory at Leningrad makes asbestos-cement products, fabrics, paper, and insulating materials. As a result of research conducted at Leningrad by the Asbestos Institute several new products and processes of manufacture have been developed.

There are several factories in Poland, Hungary, Spain, and Switzerland. Several establishments in Japan make asbestos textiles, insulating materials, and asbestos-cement products. Evidently roofing is becoming an important branch of the industry in that country, because recently there has bean an unusually large demand for shingle-stock fiber.

FABRICATION OF ASBESTOS PRODUCTS

The manufacture of asbestos products involves many diversified and complex processes. Only a brief summary of the principal operations is presented herein because this report pertains primarily to raw materials.

Fabrics. - Crude fiber delivered to the manufacturing plant usually is crushed in a chaser mill, fiberized, and freed from rock impurities by screening and air separation. It is then ready to enter the various successive stages of manufacture into fabrics. Milled spinning fibers prepared by producers need not undergo this preliminary treatment.

The processes involved in the manufacture of fabrics follow in general those employed in spinning and weaving cotton, wool, or silk. Asbestos fibers differ mainly from those of organic origin in the nature of the surface. Wool fibers are covered with scaly bands known as imbrications, and cotton fibers are rough, twisted, and irregular. On the other hand, asbestos fibers have no nodules, twists, or irregularities on the surface that will enable one fiber to cling to another. They are more nearly akin to silk but are smoother and more rodlike. This smooth, slippery condition creates such difficulties in spinning that the manufacture of a 100-percent asbestos yarn is slow and costly; therefore a percentage of some other fiber, usually cotton, is added to act as a vehicle to carry the asbestos through the manufacturing processes. The proportion of cotton added varies with the character of the asbestos and with the nature of the finished product. It rarely exceeds 20 percent and with long-fibered Canadian asbestos may be as low as 2 percent.

Asbestos used in the manufacture of fabrics may consist of a mixture of varieties from different localities. The proportions depend upon the cost and quality of raw materials and the requirements of the finished product. Most manufacturers in America prefer Canadian fiber as a base, although many products now are made of African fiber alone. The blended fibers, together with the necessary addition of cotton, are mixed thoroughly with revolving beaters. Some manufacturers, however, introduce the cotton at a later stage.

Carding is the next step in the process of manufacture. Carding rolls are covered with leather and fitted with sharp steel bristles. They comb the fibers parallel and remove short fibers, bit of rock, and dust. After passing a succession of carding rolls the fiber emerges as a loose blanket, which may be turned 90° and passed through another carding machine. The blanket then is separated into rovings, which are gathered in a roll on a Jack spool and spun into yarn, as in ordinary textile mills.

Yarns are made in various sizes; a "5-cut" yarn measures about 500 yards to the pound and a "30-cut" yarn about 3,000 yards to the pound. When twisted exceptionally hard it is known as asbestos thread and used in sewing gas mantles, asbestos theater curtains, and asbestos clothing. The spindles of single-ply yarn are transferred to twisting machines and twisted into 2- or 3-ply yarn, which is wound on spools. Asbestos cord and rope are made by twisting together a greater number of strands.

Where yarn is to be used for brake bands or packings it usually is reinforced with fine copper, brass, or lead wire. Thus, for brake bands 3 strands of single-ply yarn and 2 strands of brass wire of gage nos. 0.006, 0.007, or 0.008 may be twisted together. For packings a single lead wire or 1 to 3 strands of brass wire are twisted with 2 or 3 strands of asbestos yarn. Products prepared in this way are known as "metallic yarns."

Yarns are woven into fabrics by well-known processes employed in cotton or woolen-textile mills. Asbestos cloth is used for theater curtains, fireproof clothing, and many other textile products. Single-ply asbestos yarn is also braided into tape. For electrical insulation it should contain not more than 7 percent carbon and not more than 14 percent cotton. Metallic yarn containing about 16 percent cotton is woven into strips for brake-band linings. Standard widths range from 1 to 6 inches and standard thicknesses from 1/8 to 3/8 inch. They are processed with rubber and other ingredients. The manufacture of brake-band linings for automobiles is the most important branch of the asbestos-products industry. As indicated in table 4, strips woven for this purpose in 1929 totaled more than 20,000 miles.

Asbestos packings are made in various ways. The yarn may be twisted or braided into valve-stem packings, the braided forms may be compressed into rings, or asbestos cloth may be cut into gaskets or other desired forms. They may be coated or impregnated with rubber compounds, oil, or flake graphite. Metallic yarn is used in some packings.

Shingles and lumber. - Roofing shingles are made of a mixture of approximately 75 percent portland cement and 15 percent shingle - grade asbestos. When manufactured by the so-called "dry process" the cement, asbestos, and coloring agent are mixed dry in a cylindrical mixer provided with paddles. The mixture is spread evenly on an 18-inch conveyor belt and sprayed with water at 180°F. Rollers compress it to the required thickness, and a rotary cutter separates it into individual shingles. Red, green, or blue-black slate granules may be sprinkled on the surface and rolled in lightly. The shingles are piled in stacks separated by steel pallets and squeezed in a hydraulic press

at a pressure of 20,000 pounds per square inch, after which they are cured, trimmed, and punched for nailing.

Sheets of asbestos lumber up to 3 inches in thickness and 42 inches in width are made in the same way, but more time is required for compression and curing. Corrugated sheets are made by crimping flat sheets.

When the wet method is used cement shingle fiber and coloring matter are mixed with a large quantity of water, agitated thoroughly with a beater, and pumped to a paper machine, where sheets are built up to the desired thickness in successive laminations. Shingles thus formed are processed in the same way as those made by the dry method. Asbestos lumber is made by building to greater thickness and in larger sheets. Eternit products, both shingles and lumber, manufactured on a large scale in Austria since 1900 and during later years in the United States, are made by this process.

Paper. - Asbestos of paper-stock grade is mixed with a large amount of water to make a thin slurry, which is agitated thoroughly by 5-foot drums covered with slats. Starch, flour, or size and sodium silicate, derived partly from the overflow squeezed out of the paper at a later stage, are added to the slurry. This is then conveyed to a paper machine similar to that used in the manufacture of paper from rags or wood pulp. All particles of stone or other impurities are eliminated in a sand-catching and knot-removing machine. The sheets of paper pass between rollers to remove most of the water and are dried on hot cylinders and wound in rolls. If a 2-ply paper is desired 1 side of a sheet is coated lightly with sodium silicate, and the 2 sheets are run together over several hot rolls. Crimped paper is made by passing it over corrugated rolls. In the manufacture of air-cell pipe covering the tips of the corrugations are coated with sodium silicate, and a flat sheet is added. When this process is repeated a 2-ply, 3-ply, or thicker air-cell covering may be made. Asbestos paper has many other uses.

Compounded packings. - Certain types of asbestos packing are made by mixing clean, well-opened fiber with fillers, such as clay, barite, magnesia, iron oxide, graphite, or cellulose, and binding materials, such as gums, resins, lac, or rubber dissolved in benzine. High-quality sheets contain about 2 parts asbestos to 1 part filler. The mixture is molded into sheets of the desired thickness. Several sheets may be glued together and reinforced with copper or lead foil.

Magnesia insulation. - For the so-called "85-percent-magnesia" pipe covering a good grade of asbestos ranging from 0-6-6-4 to 0-5-8-3 is used - either chrysotile, blue asbestos, or amosite. Approximately 15 percent of fiber is mixed with 85 percent of basic magnesium carbonate derived from dolomite, magnesite, or bitterns at saltworks. These ingredients are mixed thoroughly with water and filter-pressed into cakes, which are dried for 3 days in steam-heated kilns. The blocks are then trimmed, cut in lengths, and bored to form hollow cylinders; these are sawed lengthwise, and the half-cylinder sections are rolled in pairs in glued canvas jackets. For ordinary use the walls have a maximum thickness of about 2-1/2 inches.

Asbestos cement. - A covering used widely for boiler insulation consists of short-fiber asbestos and a cementing material, such as plastic clay. The ingredients are mixed with water to form a paste which is applied with a trowel.

Cold-molded articles. - Increasing quantities of short-fiber asbestos are used in the manufacture of electrical fittings and household appliances. Mixtures of asbestos, gilsonite, cement, and oil are ground together and compressed in molds; these are baked in ovens, polished, and lacquered. Gilsonite imparts a brown color; if gray is desired the gilsonite is omitted.

Noncorrosive filters. - Amphibole asbestos, which in general is more resistant to chemicals than chrysotile, is washed, thoroughly fiberized, and otherwise prepared for use in Gooch crucibles or for other filtering processes employing strong acids and alkalies.

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COAL-MINE EXPLOSIONS AND FIRES IN THE UNITED STATES DURING THE FISCAL YEAR ENDED JUNE 30, 1935[1]

By D. Harrington[2] and W. J. Fene[3]

INTRODUCTION

The experience of the past 2 years has demonstrated clearly that mine explosions with heavy loss of life can be prevented. The investigations of the Safety Division of the United States Bureau of Mines disclose that during the fiscal year 1934 there were only 28 deaths from mine explosions in the United States, and during the fiscal year 1935 there were 54 deaths, a very favorable reduction when compared with an average death toll of 265 from this cause during the 20-year period prior to 1929. Although the frequency of occurrence of explosions has not decreased greatly during the past 7 years, there has been a marked decrease in severity, indicating that the coal-mining industry is taking more effective precautions to prevent the spread of explosions. A number of large coal-producing States, which in past years have experienced numerous disastrous explosions, have been free from such disasters for continuous periods of as much as 5 years or even longer.

A study of several hundred mine explosions occurring in the United States during the past 10 years or more indicates that the factors mainly responsible for these explosions are inadequate or defective ventilation and disregard of the hazards from explosive gas. Both of these factors are subject to control, and if controlled it is safe to say that deaths from explosions would be reduced to a negligible number as relatively few explosions are now initiated by dust.

SUMMARY OF MINE EXPLOSIONS BY STATES

Table 1 shows the results of studies of 23 explosions in 8 States, with a total of 54 deaths, during the fiscal year ended June 30, 1935; this tabulation gives data on all explosions in mines in the United States that were called to the attention of the Safety Division of the U.S. Bureau of Mines

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1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6870."

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during the fiscal year. The 23 ignitions occurring during the year are fewer than for the previous fiscal year, when 26 ignitions occurred; however, the number of deaths in 1935 was nearly twice as many - 54 compared with 28 during 1934 - indicating that there may have been some "let-down" in vigilance in safeguarding our coal operations in the past fiscal year; it certainly is to be hoped that the present fiscal year does not have the unhappy record of virtually doubling the deaths from explosions, as was the case in the fiscal year just ended compared with the previous fiscal year.

The States reporting the greatest number of deaths from explosions were: Pennsylvania, 24 (18 in anthracite mines); Virginia, 18; West Virginia, 5; Oklahoma and Wyoming, 3 each; and Kentucky, 1. Pennsylvania also led in number of ignitions with 11 (8 in anthracite mines); West Virginia was next with 4; Oklahoma and Virginia had 2 each; Alabama, Kentucky, Wisconsin (metal mine), and Wyoming had 1 each.

Two major explosions occurred during the fiscal year; one was in an anthracite mine and resulted in 13 deaths and one in a bituminous mine, with 17 deaths. During the previous fiscal year there was but 1 major explosion, with 7 deaths, and the year before that there were 3 major explosions, with 91 deaths.

Twenty-nine persons were killed during the past fiscal year in explosions caused by electricity compared with an average of 19 deaths per year for the preceding 3 years. There were 10 electrical ignitions during each of the last 2 years; however, a larger number of deaths per ignition was recorded in the past fiscal year.

West Virginia had the greatest number of electrical ignitions - 3, with 4 fatalities; Virginia had 2, with 18 fatalities; Pennsylvania, 2, with 3 fatalities; Wyoming, 1, with 3 fatalities; Kentucky, 1, with 1 fatality; and Alabama, 1, which resulted in 4 injuries but no fatalities.

Nine ignitions were caused by open lights or matches (5 of which were in anthracite mines) compared with 9 in 1934 and an average of 10 during the previous 6 fiscal years. Six deaths resulted from explosions caused by open lights and matches last year compared with an average of 33 during the previous 6 fiscal years. Ignitions by open lights and matches were distributed as follows: Pennsylvania, 5, with 2 fatalities; Oklahoma, 2, with 3 fatalities; West Virginia, 1, with 1 fatality; and Wisconsin (metal mine), 1, with no fatalities. Fewer deaths have been caused by open lights and matches during the past 2 years than in any year in the history of coal mining in the United States; this record probably is due to the fact that most of the mines of the United States which are heavy producers of explosive gas are now equipped with electric cap lamps.

TABLE 1. - Summary of mine explosions by States, July 1, 1934, to June 30, 1935

State	Lights				Fatalities by ignition causes									
	Open	Closed	Unknown or no lights	Total mine explosions	Electricity		Open lights or smoking		Explosives		Unknown		Total	
					Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions
Alabama	-	1	-	1	-	1	-	-	-	-	-	-	-	1
Kentucky	1	-	-	1	1	1	-	-	-	-	-	-	1	1
Oklahoma	1	1	-	2	-	-	3	2	-	-	-	-	3	2
Pennsylvania	3	8	-	1/ 11	3	2	2	5	3	2	16	2	24	11
Virginia	-	2	-	2	18	2	-	-	-	-	-	-	18	2
West Virginia	-	4	-	4	4	3	1	1	-	-	-	-	5	4
Wisconsin	2/ 1	-	-	2/ 1	-	-	-	2/ 1	-	-	-	-	-	2/ 1
Wyoming	-	1	-	1	3	1	-	-	-	-	-	-	3	1
Total, 1935	6	17	-	23	29	10	6	9	3	2	16	2	54	23
Total, 1934	9	14	3	26	21	10	2	9	2	3	3	4	28	26
Total, 1933	7	14	1	22	25	8	63	8	29	4	5	2	122	22
Total, 1932	2/ 15	2/ 15	2/ 4	2/ 34	11	10	33	15	42	5	1	3	87	33
Total, 1931	12	15	-	26	143	9	56	11	18	6	-	-	217	26
Total, 1930	8	25	1	34	154	18	21	8	24	6	-	2	199	34
Total, 1929	18	19	1	38	93	18	14	11	26	8	6	1	139	38

1/ 7 of these explosions were in anthracite mines.
2/ Ignition of methane in metal mine.

Note. - Explosions are recorded by fiscal years, and totals are for the United States.

Two explosions with 3 fatalities originated from explosives during the past fiscal year compared with 3 explosions and 2 deaths from this cause during the fiscal year 1934 and an average of 10 explosions and 47 deaths for the past 6 years; this is also a new low record for explosions caused by explosives - one to be commended, as explosions from the use or rather the misuse of explosives are caused very easily. Both of these explosions occurred in Pennsylvania; one in an anthracite mine was caused by a mud-cap shot, and one in a bituminous mine was thought to have been caused by the accidental ignition of the explosives while being handled.

Two explosions in which the cause of ignition was not ascertained definitely occurred in 2 Pennsylvania anthracite mines, one resulting in 13 deaths and the other in 3 deaths. The one killing 13 is believed to have been caused by blasting; if this explosion is assessed against blasting the death record from such explosions would still be lower than for several years past with the exception of the fiscal year 1933-34, which had a remarkably low death rate from this cause.

RECORD OF EXPLOSIONS BY STATES

Table 2 records explosions during the past 7 fiscal years by States. During the 7-year period 202 ignitions caused 846 deaths, a yearly average of 28.86 ignitions and 120.86 deaths. The table lists only the explosions with which the Safety Division had contact; nevertheless it probably includes at least 98 percent of the explosions in the United States in which there was loss of life. The number of both ignitions and deaths during the past year was considerably less than the average for the 7-year period; there were also 3 fewer ignitions and 26 more deaths (23 ignitions and 54 deaths) than in the previous year (26 ignitions and 28 deaths). For the 7-year period (July 1, 1928 to June 30, 1935) Pennsylvania heads the list in total number of ignitions (61) and deaths (165); Oklahoma with but 10 ignitions ranked second in number of deaths (123), with no ignitions during 2 years of the period. It is of interest to note that during the 7 years Maryland, Wisconsin, Kansas, and Wyoming had 1 ignition each with a total of 6 deaths; that Tennessee had 3 ignitions without a fatality; and that Ohio had no ignitions in 4 years and Utah none in 5 years. The total average number of deaths per ignition is 4.21, the various States ranging from none to 11.50 deaths per ignition. No deaths resulted from 7 of the 23 explosions during the last year. The following coal-mining States and Territories are not listed in table 2 as having had any explosion during the past 7 fiscal years: Alaska, Arkansas, Iowa, Michigan, Missouri, Montana, North Dakota, and Texas.

[illegible]

[illegible]

[illegible]

[illegible]

TABLE 2. - Record of explosions by States during last 7 fiscal years

State	1929		1930		1931		1932		1933		1934		1935		Total 7 years		Average per year		Average deaths per ignition
	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions	
Alabama	13	7	8	4	-	2	4	5	-	-	2	2	-	1	27	21	3.86	3.00	1.29
California	-	1	7	2	12	1	-	1	-	1	-	-	-	-	19	6	2.71	.87	3.17
Colorado	-	-	-	1	-	-	4	3	1	1	-	-	-	-	5	5	.71	.71	1.00
Illinois	-	3	10	2	2	1	3	3	54	1	-	-	-	-	69	10	9.86	1.43	6.90
Indiana	-	-	-	-	28	1	-	-	4	2	-	-	-	-	32	3	4.57	.43	10.66
Kansas	2	1	-	-	-	-	-	-	-	-	-	-	-	-	2	1	.29	.28	2.00
Kentucky	11	3	16	1	-	-	3	1	26	2	-	1	1	1	57	9	8.14	.87	6.33
Maryland	-	-	-	-	-	-	1	1	-	-	-	-	-	-	1	1	.14	.14	1.00
New Mexico	-	-	-	-	8	3	-	-	14	1	-	-	-	-	22	4	3.14	3.14	5.50
Ohio	3	3	1	2	88	3	-	-	-	-	-	-	-	-	92	8	13.14	1.14	11.50
Oklahoma	1	1	71	4	45	2	-	-	3	2	-	-	3	2	123	11	17.57	1.57	11.18
Pennsylvania	70	9	16	8	23	7	7	6	9	6	16	14	24	11	165	61	23.57	8.71	2.75
Tennessee	-	-	-	-	-	-	-	3	-	-	-	-	-	-	-	3	---	.43	---
Utah	1	2	30	3	-	-	-	-	-	-	-	-	-	-	31	5	4.43	.71	6.20
Virginia	2	1	-	-	-	-	54	3	-	-	2	3	18	2	76	9	10.87	1.29	8.44
Washington	-	-	17	2	-	2	-	-	2	1	-	2	-	-	19	7	2.71	1.00	2.71
West Virginia	36	7	23	5	11	4	11	7	9	5	8	4	5	4	103	36	14.57	5.14	2.86
Wisconsin	-	-	-	-	-	-	-	-	-	-	-	-	-	1	-	1	---	.14	---
Wyoming	-	-	-	-	-	-	-	-	-	-	-	-	3	1	3	1	.43	.14	3.00
Total	139	38	199	34	217	26	87	33	122	22	28	26	54	23	846	202	120.86	28.86	4.21

CAUSES OF EXPLOSIONS

Table 3 lists causes of explosions in the United States during the past 7 fiscal years, as reported to the Safety Division of the United States Bureau of Mines. It will be noted that 202 explosions have been studied during the 7-year period, an average of approximately 28.8 a year. There were 23 explosions during the past fiscal year, 3 less than during the previous year and 5.8 less than the yearly average for the period. Eighty-three (41.1 percent) of these explosions (an average of 12 a year) during the 7-year period were of electrical origin; this source caused 43.5 percent of the explosions during the past fiscal year, a 5-percent increase over the preceding year and a 2-percent increase over the average for the 7-year period.

Table 3 shows that explosives were responsible for 8.7 percent of the explosions during the past fiscal year compared with 11.5 percent for the previous year and with an average of 16.9 percent for the 7-year period. Undoubtedly numerous explosions have been caused by explosives during the past 7 years in addition to the 34 listed in table 3, as many explosions originating from blasting which do not result in deaths or serious property damage may not come to the attention of the Safety Division personnel. The reduction in explosions from explosives may be attributed chiefly to safer practices in the handling of explosives and to the increased use of permissible explosives.

Open lights or smoking were responsible for 9 (39.1 percent) of the explosions occurring during the past fiscal year, the same number as in the previous year and slightly below the average for the 7-year period. Three of these explosions occurred when firebosses attempted to relight their safety lamps with matches; this practice has caused several disastrous explosions in past years. As it is impossible to control the actions of men completely key-locked safety lamps should be barred from every coal mine.

The cause of two of the explosions (8.7 percent of the total) was not ascertained. Both explosions listed under "unknown cause" were in anthracite mines; one resulted in 13 deaths and 50 injuries and the other in 3 deaths and 6 injuries; the first is believed to have been started by blasting.

EXPLOSION FATALITIES BY CAUSES

Table 4 compares explosion fatalities by causes and shows that during the past 7 fiscal years 846 fatalities occurred from explosions in mines of the United States, a yearly average of 120.8. The record of 1934, when 28 were killed by explosions, is by far the best of any one year in the mines of the United States since records have been available; the record of the fiscal year 1935, when 54 were killed, is the next best.

TABLE 3. - Comparison of explosion - ignition causes, fiscal years 1929 to 1935, inclusive

Fiscal year	Ignition by electricity		Ignition by open lights or smoking		Ignition by explosives		Ignition by unknown causes		Total	
	Number	Percent	Number	Percent	Number	Percent	Number	Percent	Number	Percent
1929	18	47.4	11	29.0	8	21.0	1	2.6	38	100
1930	18	52.5	8	23.5	6	18.0	2	6.0	34	100
1931	9	34.6	11	42.3	6	23.1	-	.0	26	100
1932	10	30.1	15	45.5	5	15.2	3	9.2	33	100
1933	8	36.4	8	36.4	4	18.1	2	9.1	22	100
1934	10	38.5	9	34.6	3	11.5	4	15.4	26	100
1935	10	43.5	9	39.1	2	8.7	2	8.7	23	100
Total, 7 years	83		71		34		14		202	
Average, 7 years	11.8	41.1	10.1	35.1	4.9	16.9	2.0	6.9	28.8	100

TABLE 4. - Comparison of explosion fatalities by ignition causes, fiscal years 1929 to 1935, inclusive

Fiscal year	Electricity		Open lights or smoking		Explosives		Unknown		Total	
	Number killed	Percent	Number killed	Percent	Number killed	Percent	Number killed	Percent	Number killed	Percent
1929	93	66.9	14	10.1	26	18.7	6	4.3	139	100
1930	154	77.5	21	10.5	24	12.0	-	---	199	100
1931	143	65.9	56	25.8	18	8.3	-	---	217	100
1932	11	12.7	33	38.0	42	48.2	1	1.1	87	100
1933	25	20.5	63	51.6	29	23.8	5	4.1	122	100
1934	21	75.0	2	7.1	2	7.1	3	10.8	28	100
1935	29	53.7	6	11.0	3	5.6	16	29.7	54	100
Total, 7 years	476		195		144		31		846	
Average 7 years	68.0	56.3	27.6	23.0	20.7	17.0	4.4	3.7	120.8	100

Explosions of electrical origin accounted for 56.3 percent of the deaths from explosions during the past 7 fiscal years; during the past year 53.7 of the fatalities from explosions - a little less than the average - were of electrical origin.

The fact that electricity continues to lead the list as a cause of explosions in mines evidences a disregard of the hazard from electric arcs in the presence of gas; there is good reason to believe that if rock-dusting had not been in effect the fatalities from explosions of such origin would be numerous.

Open lights or smoking caused explosions which resulted in 195 deaths during the past 7 fiscal years or 23 percent of the total fatalities from explosions during the period. During the past year 6 deaths (11 percent of the total) were attributed to this cause compared with 2 deaths in 1934. Five of the mine explosions caused by open lights or smoking during the year were in anthracite mines; two of the explosions occurred when men attempted to smoke in the presence of gas. Such carelessness shows the necessity of rigid discipline in this respect; anthracite mines are far more guilty of this breach of discipline and commonsense mine-safety precautions than bituminous mines.

During the past 7 years 144 deaths (17 percent of the total killed during this period from all causes) resulted from explosions caused by the misuse of explosives. Explosions caused by blasting have been reduced markedly during the past 2 years; only 5 deaths were reported during these 2 years compared with a yearly average of 20.7 deaths. One of the explosions caused by blasting during the fiscal year resulted from a "mud-cap" shot, and the other was thought to have resulted from careless handling of explosives. If one explosion with 13 fatalities, listed as of unknown origin but believed to have been caused by blasting, should be assessed against explosions caused by explosives the number would be 16 instead of 3 but it would still be lower than the 7-year average.

During the past 7 years 31 persons were killed in the mines of the United States in 14 explosions, the causes of which could not be ascertained by the Bureau of Mines Safety Division. It is often difficult to determine the cause of an explosion, because usually the persons involved are killed and the surrounding conditions are changed. During the past fiscal year 2 of the explosions resulting in 16 deaths were of unknown origin, both occurring in anthracite mines.

ELECTRICAL CAUSES OF EXPLOSIONS

Table 5 lists causes of explosions of electrical origin occurring in the United States during the past 7 fiscal years, insofar as data are available to the Safety Division of the Bureau of Mines. During the 7-year period there were 83 ignitions of electrical origin and during the past fiscal year, 10 ignitions with 29 fatalities. The greatest sources of electrical ignition in coal mines are trolley or cable-reel locomotives and nonpermissible mining

machines; these two are responsible for approximately half of the electrical ignitions that occurred during the 7-year period. During the past fiscal year trolley or cable-reel locomotives accounted for the greatest number of ignitions, but nonpermissible electric drills accounted for the greatest number of deaths. Nonpermissible mining machines and nonpermissible electric pump motors were each responsible for two ignitions.

In most of the electrical ignitions there was evidence of carelessness in maintaining good ventilation or adequate rock-dusting in the sections in which the ignitions occurred. Other factors contributing to these electrical ignitions were: Failure to test for gas before going into working place with mining machine or drill; use of electrical equipment too soon after mine fan had been started; and use of haulageways as return airways.

COMPARISON OF LIGHTING IN MINES WHERE EXPLOSIONS OCCURRED

In a comparison of lighting in mines where explosions occurred during the past 7 fiscal years it is observed that of the 204 explosions listed 75 (36.8 percent of the total) occurred in open-light mines; that 119 (58.3 percent) were in closed-light mines; and that in 10 (4.9 percent) the lighting practice was unknown. The fact that the majority of these explosions occurred in closed-light mines indicates that far too much reliance is placed on closed-light equipment with either lack of appreciation of the hazards of inadequate ventilation, nonpermissible electrical equipment, and nonpermissible explosives or only a half-hearted attempt to control ignitions of gas or dust. No explosion was initiated by ignition of gas or dust by an electric cap lamp, though unquestionably some have been initiated by flame safety lamps either through use or misuse. Some coal-mining people are inclined to throw far too much burden on the well-established safety dependability of electric cap lamps and are using the fact that the electric cap lamp is safe equipment to cover the numerous unsafe features of nonpermissible electrical equipment, nonpermissible explosives, and defective or neglected ventilation.

TABLE 5. - Electrical causes of explosions, fiscal years 1929 to 1935, inclusive

Cause	Number of electrical ignitions by fiscal years							Total, 7 years	Number killed		Number injured	
	1/1929	2/1930	3/1931	4/1932	5/1933	6/1934	1935		1934	1935	1934	1935
Trolley or cable-reel locomotive	7/5	5	1	1	4	5	3	7/23	13	1	4	4
Nonpermissible mining machine	8/5	5	3	3	1	-	2	8/18	—	4	-	4
Nonpermissible electric motor	4	3	-	-	-	-	2	9	—	2	-	-
Nonpermissible shotfiring battery or circuit	-	9/ 2	9/ 3	1	-	1	-	2	—	-	3	-
Cable nips or blow-outs	-	2	-	-	-	1	-	3	1	-	3	-
Nonpermissible electric drills	-	-	-	2	-	1	2	5	2	20	3	12
Nonpermissible storage-battery locomotives	1	1	2	-	-	-	-	4	—	-	-	-
Nonpermissible electric hoists	-	-	-	-	1	-	-	1	—	-	-	-
Faulty wiring	2	2	-	-	-	-	-	4	—	-	-	-
Trolley wire	1	-	2	1	-	-	-	4	—	-	-	-
Electric arc	2	-	1	2	1	2	1	9	5	2	7	-
Signal wire	-	-	-	-	1	-	-	1	-	-	-	-
Total ignitions	18	18	9	10	8	10	10	83	21	29	20	20

1/ See Inf. Circ. 6178. 2/ See Inf. Circ. 6419. 3/ See Inf. Circ. 6540. 4/ See Inf. Circ. 6680. 5/ See Inf. Circ. 6761. 6/ See Inf. Circ. 6819.
7/ 1 not included in totals; may have been open light, hence is assigned to open lights or smoking.
8/ 1 not included in totals; may have been from smoking, hence is assigned to open lights or smoking.
9/ Not included in totals; may have been from overcharged shots, hence is assigned to explosives accidents.

TABLE 6. - Comparison of lighting in mines where explosions occurred, fiscal years 1929 to 1935, inclusive

Fiscal year	Open lights		Closed lights		Lighting unknown		Total	
	Number of explosions	Per-cent	Number of explosions	Per-cent	Number of explosions	Per-cent	Number of explosions	Per-cent
1929	18	47.4	19	50.0	1	2.6	38	100
1930	8	23.5	25	73.5	1	3.0	34	100
1931	12	44.4	15	55.6	-	---	1/ 27	100
1932	15	44.1	15	44.1	4	11.8	1/ 34	100
1933	7	31.8	14	63.6	1	4.6	22	100
1934	9	34.6	14	53.9	3	11.5	26	100
1935	6	26.1	17	73.9	-	---	23	100
Total, 7 years	75	36.8	119	58.3	10	4.9	2/204	100

1/ 1 mine had mixed (open as well as closed) lighting.
2/ 2 mines had mixed (open as well as closed) lighting.

MINE FIRES

Although mine fires, especially those in coal mines, are potential causes of great loss of life and tremendous property damage the mines of the United States have been fortunate during the past decade in that they have not experienced a fire that caused large loss of life; however, a number of fires have involved considerable property damage, and some mining companies persist in using methods or equipment that cause mine fires several times essentially every month throughout the year.

During the past fiscal year 26 mine fires were called to the attention of the United States Bureau of Mines Safety Division. No doubt, scores of other fires were detected and extinguished promptly without much damage or were handled quietly and more or less secretly by the mine involved; hence they were not made public.

Table 7 shows that Kentucky had the most fires in the fiscal year 1935; 6 fires were reported in that State during the year. Three fires each were reported in Colorado, Pennsylvania, and West Virginia; 2 in California; and 1 each in Alabama, Michigan, Ohio, Oklahoma, Tennessee, Utah, Virginia, Washington, and Wyoming. Five of these 26 fires were in metal mines. Electricity was responsible for 11 of the fires and 11 deaths; in 1 shaft fire 6 men were killed in an attempt to control the fire. Open lights started one of the fires in which hay was ignited in a mule barn. Of the 14 fires listed under miscellaneous 6 are of unknown origin; 4 probably were caused by spontaneous combustion; 1 was probably of incendiary origin; 1 was started by a forest fire which ignited drift timbers; 1 was caused by a lighted cigarette; and 1 was a fire that has been burning about 50 years and has covered an area of about 6 square miles. The total number of fires (26) during the fiscal year 1935 is above the average (22.8) for the preceding 5 years; also, the number of deaths (11) due to fires is an increase over the preceding 3 years.

TABLE 7. - Summary of mine fires by States, July 1, 1930, to June 30, 1935

State	Lights				Fatalities by ignition causes									
					Electricity		Open lights		Explosives		Miscellaneous		Total	
	Open	Closed	Unknown	Total mine fires	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions	Deaths	Igni-tions
Alabama	-	1	-	1	-	1	-	-	-	-	-	-	-	1
California	2	-	-	2	-	-	-	1	-	-	-	1	-	2
Colorado	1	1	1	3	1	1	-	-	-	-	-	2	1	3
Kentucky	3	-	3	6	-	4	-	-	-	-	-	2	-	6
Michigan	-	-	1	1	1	1	-	-	-	-	-	-	1	1
Ohio	-	-	1	1	-	-	-	-	-	-	-	1	-	1
Oklahoma	-	-	1	1	1	1	-	-	-	-	-	-	1	1
Pennsylvania	-	-	3	3	-	-	-	-	-	-	-	3	-	3
Tennessee	-	-	1	1	-	-	-	-	-	-	-	1	-	1
Utah	-	-	1	1	-	-	-	-	-	-	-	1	-	1
Virginia	-	1	-	1	2	1	-	-	-	-	-	-	2	1
Washington	-	-	1	1	-	-	-	-	-	-	-	1	-	1
West Virginia	1	2	-	3	6	2	-	-	-	-	-	1	6	3
Wyoming	-	-	1	1	-	-	-	-	-	-	-	1	-	1
Total, 1935	7	5	14	26	11	11	-	1	-	-	-	2/14	11	1/26
Total, 1934 3/	10	5	3	18	-	4	-	-	5	5	1	9	6	18
Total, 1933 4/	12	6	9	27	-	7	-	-	-	2	2	18	2	27
Total, 1932 5/	8	5	8	21	1	8	-	-	-	3	-	10	1	21
Total, 1931 6/	12	7	5	24	-	7	5	3	-	5	18	9	23	24
Total, 1930 7/	7	8	9	24	2	9	3	6	-	3	-	6	5	24

1/ Includes 5 metal-mine fires and 4 outcrop fires.
2/ 4 of these fires were caused by spontaneous combustion.
3/ See Inf. Circ. 6819
4/ See Inf. Circ. 6761.
5/ See Inf. Circ. 6680.
6/ See Inf. Circ. 6540.
7/ See Inf. Circ. 6419.

Note. - This table includes data only on mine fires to which the attention of the Bureau of Mines Safety Division was called; mine fires are recorded by fiscal years.

SUMMARY

It is encouraging to note that during the past 7 years the trend in the frequency and severity of explosions in the mines of this country has been definitely downward. Although the number of deaths from explosions during the fiscal year 1935 was nearly double that for 1934 (which was the best record in the history of coal mining in the United States) the record is still below the average for the 5 years preceding 1934; because of the numerous abnormal conditions and increased mining activities in a number of the important coal-producing States, some mining people feared that fires and explosions would be frequent, but fortunately that fear was not realized.

The years 1918, 1920, 1921, 1931, 1933, and 1934 have the best records as regards major fires and explosions in the mines of the United States; in the last year only 3 major disasters occurred, in all killing 36 men, and in the previous year (1933), only 1, killing 7 men. The material decline in frequency and severity of explosions and fires during the past 4 years is believed to be due primarily to: (1) Increased activities of the various State mine-inspection departments, especially in educational work and demand for more rigid adherence to State laws and safety practices far beyond the strict letter of the law; (2) the growing realization of the economic waste of all accidents, including explosions, and the more or less general acceptance of responsibility for accident occurrence by the mine operators; (3) the increased interest in accident prevention by the workers; and (4) the widespread dissemination of safety data by the United States Bureau of Mines and other safety organizations.

It is definitely known that mine explosions and fires can be prevented by the adoption of certain standard preventive measures, the expense of which is negligible compared with probable ultimate savings. The adoption of these preventive measures is largely a matter of the education of miners and operators to the seriousness of existing or potential hazards and a realization by the mine operators that most of these "accidents" can be prevented and that much of the burden rests squarely on their shoulders.

Table 8 presents data on major (those in which 5 or more lives were lost) coal-mine fire disasters in the United States since 1900. It is interesting to note that of the 24 major coal-mine fire disasters with a total of 552 fatalities since 1900, only 2 with 15 deaths have occurred since 1921 and that no major coal-mine fire disasters occurred in the United States during the period 1926-33, inclusive.

TABLE 8. - Major coal-mine fires in the United States during the past 34 calendar years in which 5 or more were killed

Year	Number of fires	Killed
1901	2	36
1902	1	10
1903	0	-
1904	1	5
1905	3	19
	7	70
1906	1	8
1907	1	5
1908	2	38
1909	2	268
1910	1	10
	7	329
1911	1	72
1912	1	9
1913	0	-
1914	1	5
1915	0	-
	3	86
1916	0	-
1917	0	-
1918	1	13
1919	1	20
1920	1	6
	3	39
1921	2	13
1922	0	-
1923	0	-
1924	0	-
1925	1	9
	3	22
1926	0	-
1927	0	-
1928	0	-
1929	0	-
1930	0	-
	0	-
1931	0	-
1932	0	-
1933	0	-
1934	1	6
	1	6
Grand total, 1901-34	24	552

Some of the salient points brought out in the study of mine explosions and fires in mines of the United States by the Safety Division of the United States Bureau of Mines for the year ended June 30, 1935 are as follows:

1. Twenty-three explosions were studied, 16 in bituminous mines and 7 in anthracite mines, compared with 26 for the previous year and a yearly average of 29.8 for the previous 6 fiscal years.

2. Explosions took 54 lives in the last fiscal year compared with 28 the previous fiscal year and an average of 132 for the previous 6 fiscal years. The 7 explosions in anthracite mines during the past year resulted in 18 deaths, whereas the 16 explosions occurring in bituminous mines resulted in 36 deaths.

3. The two major explosions during the fiscal year accounted for 30 deaths, while the 21 minor explosions accounted for 24 deaths; in 6 of the explosions no deaths resulted.

4. Six explosions occurred in open-light mines compared with 9 in 1934 and a yearly average of 11.5 for the previous 6 fiscal years. Seventeen explosions were in closed-light mines compared with 14 in 1934 and a yearly average of 17 for the previous 6 fiscal years; in three of the closed-light mines gas was ignited when firebosses attempted to relight their flame safety lamps with matches.

5. Ten explosions, causing 29 deaths, were of electrical origin compared with the same number of explosions and 21 deaths for the previous fiscal year. Explosions of electrical origin averaged 12.5 and deaths 74.5 annually for the previous 6 fiscal years. The past 4 fiscal years show a decided improvement in the occurrences of electrical explosions.

6. West Virginia, with 3 explosions, ranks first in the number of explosions caused by electricity during the year, and Virginia with 18 deaths ranks first in the number of deaths from this cause.

7. Trolley and cable-reel locomotives continue to be the greatest source of explosions of electrical origin, accounting for 3 of the electrical explosions and 1 death; nonpermissible electric drills caused 2 explosions and 20 deaths.

8. Open lights or smoking caused 9 explosions with 6 deaths, compared with 9 explosions and 2 deaths in the previous year and an average of 10.3 explosions and 31.5 deaths during the previous 6 fiscal years. Three of the explosions occurring during the fiscal year 1935 were charged to firebosses who attempted to relight flame safety lamps with matches.

9. Two explosions and 3 deaths were attributed definitely to misuse of explosives compared with 3 explosions and 2 deaths from this cause in 1934 and an average of 5.3 explosions and 23.5 deaths during the previous 6 years. The 3 deaths caused by explosives during the past year resulted from ignition of gas

by a "mud-cap" shot; in one explosion, the cause of which is unknown, strong suspicion has been directed toward blasting.

10. Rock dust was used to some extent in at least 5 mines in which explosions occurred during the past year. In two of the mines in which explosions occurred rock dust is credited with stopping the spread of the explosions.

11. Twenty of the explosions during the past year are thought to have been initiated by gas and one by coal dust.

12. The underlying cause of at least 51 of the 54 deaths resulting from explosions during the past fiscal year was defective ventilation, manifest both in open-and closed-light mines.

13. The severity of explosions last year was reduced approximately 50 percent compared to the average for the preceding 6 years. The average number of deaths per explosion last year was 2.3 compared with an average of 4.4 deaths per explosion for the preceding 6 years.

14. During the fiscal year 1935, 26 mine fires with 11 deaths were reported compared with an average of 22.8 fires and 7.4 deaths per year for the preceding 5 years.

15. The 54 deaths from explosions during the fiscal year 1935 occurred in 8 States; 19 of the coal-producing States had no fatalities from this cause.

CONCLUSIONS

The data in this circular were compiled in the same manner as in previous years (see Inf. Circs.6178, 6419, 6540, 6680, 6761, and 6819) and embody only facts revealed through studies by field employees of the United States Bureau of Mines Safety Division. Although an attempt is made to obtain information on every explosion that occurs in the United States some of the less destructive are not brought to the attention of the Bureau early enough to permit investigation; rarely, if ever, does the Bureau fail to acquire fairly complete details of mine explosions in which more than one life is lost or in which a considerable amount of property is damaged. The data in these publications are therefore relatively complete, especially those on explosions causing loss of more than one life. The data on fires are by no means as complete as those for explosions, and there is reason to believe that scores of fires in both coal and metal mines occur annually without the knowledge of the United States Bureau of Mines or even the State inspection forces. Mine fires occur almost daily in some coal mines, particularly at shot-firing time or where the coal fires spontaneously, and fires of electrical origin or from open lights are not unusual in metal mines.

A study of the data available on explosions and fires in the United States during the past 6 or 7 years reveals an encouraging reduction in the frequency and severity of these disasters.

In the 20-year period preceding 1929 mine explosions and fires took an average toll of about 285 lives annually in the United States; since then the number of fatalities from explosions and fires has been reduced to an average of 128. The fiscal year 1934 established an all-time low record - only 34 fatalities resulted from explosions and fires compared with 65 in the fiscal year 1935. There is no good reason why the good record of 1934 cannot be equaled every year; in any year in which the record is not better than in the fiscal year 1934 certainly negligence has been the cause. Mine explosions and fires are preventable and most certainly should be prevented.

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January 1936

Department of the Interior

United States Bureau of Mines
John W. Finch, Director

INFORMATION CIRCULAR

HOW TO USE PERMISSIBLE EXPLOSIVES PROPERLY

BY

D. Harrington and S. P. Howell

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DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

HOW TO USE PERMISSIBLE EXPLOSIVES PROPERLY[1/]

By D. Harrington[2/] and S. P. Howell[3/]

INTRODUCTION

Probably over 95 percent of the coal production of the United States depends upon the use of explosives before it can be placed in the railroad car at the mine, and up to the present time it has been impossible to obtain any kind of practicable explosive that isn't dangerous unless handled carefully. In other words, no "foolproof" explosives have been found to date, and inspection of the accident records of the coal mines of this country indicates that the use of explosives in coal mines has taken a sad toll of lives.

ACCIDENTS FROM EXPLOSIVES IN COAL MINES

Of the 2,000 to 2,500 persons killed annually in the coal mines of the United States, 100 to 125 have been killed from explosives without an explosion of gas or dust; these accidents have been due to misfires, hangfires, premature blasts, or accidental setting off of explosives through carelessness or otherwise (but chiefly through carelessness in some form). In this type of accident the flame character of the explosive is not necessarily involved, although an unexpected or premature blast is initiated much more easily with black blasting powder, which has a long flame, than with a permissible explosive, which has a flame much shorter and of briefer duration. By far the greater number of such accidents are caused by black blasting powder or dynamite rather than by permissible explosive.

Although the hazard from premature blasts is much greater with long-flame black blasting powder than with short-flame permissible explosives (and this alone should outlaw black blasting powder from coal mines) by far the greater hazard from long-flame explosives (black blasting powder) is found in the fact that about one third of our coal-mine explosions have been initiated by blown-out shots or other effect of misuse of explosives, and at least 90 percent of these have been due to black blasting powder. In some

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6871."

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regions with a record of numerous explosions blown-out shots (chiefly of black powder but occasionally of dynamite) have caused two thirds of the disasters. In addition, innumerable fires have been caused by black powder or dynamite shots in coal mines; one mining company operating fewer than 10 mines had a record (covering a period of several years) of more than 1 fire a week from blasting with black powder or dynamite or a mixture of the two.

Of the coal-mine explosions investigated by the United States Bureau of Mines, 1910 to November 1926, involving 144 disasters with 3,756 fatalities, 80, or about 56 percent, were from blown-out shots, and it is thought that black blasting powder was responsible for at least 70 of the 80.

The record as regards responsibility for coal-mine explosions by no means favors explosives; this fact was recognized more than 20 years ago. The problem was one of the most pressing that confronted the Bureau of Mines upon its creation in 1910; the Bureau therefore began a vigorous study of the nature of explosives and made tests to determine their suitability for use in coal mines, developing what is known as its "permissible explosives" list, though "short-flame" explosives were being used to a limited extent at least as early as 1902.

A permissible explosive is one that has passed certain tests conducted by the United States Bureau of Mines at the Explosives Experiment Station, Bruceton, Pa. After the explosive has passed these tests it is placed on the official list of explosives permissible for use in coal mines and may be sold as a permissible explosive. The word "permissible" refers not only to the explosive itself but also to the method of using the explosive underground. Thus, in addition to specifying that the explosive actually used should be similar in all respects to the sample submitted by the manufacturer for test, the Bureau specifies the use of electric detonators and states that the explosive must not be used in a frozen condition, that the quantity used for a shot must not exceed 1-1/2 pounds, and that the charge must be confined properly in a borehole with clay or other incombustible stemming. The Bureau also advocates that all coal which it is feasible to cut should be cut or sheared.

CLASSES OF PERMISSIBLE EXPLOSIVES

The Bureau of Mines classifications of permissible explosives, as well as the conditions of use to retain permissibility, as given in Report of Investigations 3220 are:

> In order that the user of explosives may be assisted in selecting an explosive to meet a specific requirement, the United States Bureau of Mines now classifies permissible explosives in two ways, as follows: (1) On the basis of the volume of poisonous gases produced by 680 grams (1-1/2 pounds) of the explosive and (2) on the basis of the characteristic ingredient of each explosive.

Volume of Poisonous Gases

Most of the permissible explosives, even when properly and completely detonated in a drill hole in a coal mine, produce poisonous gases, but they produce at the same time a much larger volume of nonpoisonous gases. In order that the poisonous gases may not under normal conditions become a menace to the lives or health of miners, no explosive is now or can become permissible if it evolves upon detonation more than 158 liters (5-1/2 cubic feet) of permanent poisonous gases, as determined by tests in the Bichel pressure gage. Field tests of an explosive made under extreme conditions for the production of the greatest percentage of poisonous gases in the air show that in a narrow entry, with no ventilation at or near the face, a 1-1/2-pound charge of an explosive, which gave 158 liters of poisonous gas in gage tests, produced 0.18 percent of carbon monoxide (the only poisonous gas present) in the air when the sample was taken 2 minutes after the shot. Another sample of the air taken 2 minutes later contained 0.08 percent of carbon monoxide. It is therefore evident that where ventilation is not active, as in a closed entry, miners or shot firers should not return to the face until at least 5 minutes after a shot. At all working faces that are difficult to ventilate explosives of class A or class B should be used, preferably those of class A.

The classification on the basis of the volume of poisonous gases produced by 680 grams (1-1/2 pounds) of the explosive is as follows:

Class A, those explosives from which the volume of poisonous gases produced is not more than 53 liters.

Class B, those explosives from which the volume of poisonous gases is more than 53 liters but less than 106 liters, inclusive.

Class C, those explosives in which the volume of poisonous gases is more than 106 liters but less than 158 liters, inclusive.

Characteristic Ingredients

Explosives are classified in accordance with their characteristic ingredients as follows:

Class 1, ammonium nitrate explosives. - To class 1 belong all the explosives in which the characteristic ingredient is ammonium nitrate. This class is divided into two subclasses. Subclass a includes every ammonium nitrate explosive that contains a sensitizer that is in itself an explosive. Subclass b includes every ammonium nitrate explosive that contains a sensitizer that is not in itself an explosive. The ammonium nitrate

explosives of subclass a consist principally of ammonium nitrate with small percentages of nitroglycerin, nitrocellulose, or nitro-substitution compounds which are used as sensitizers. The ammonium nitrate explosives of subclass b consist principally of ammonium nitrate with small percentages of resinous matter or other nonexplosive substances used as sensitizers.

Ammonium nitrate explosives when fresh and properly detonated are well-adapted for use in mines that are not unusually wet. They are not suitable for use in wet mines; for if the contents of a cartridge of ammonium nitrate explosive is exposed for only a few hours to the damp atmosphere the explosive may so deteriorate as to fail to detonate completely, because ammonium nitrate takes up moisture readily. The redipping of cartridges of ammonium nitrate explosives aids in protecting the contents against moisture, or moist air, and the cartridges should be so stored and handled as to preserve the efficacy of the paraffinlike coating. The explosives should be obtained in a fresh condition and purchased in such quantities as will permit their prompt use. Experience at the Pittsburgh Experiment Station of the United States Bureau of Mines shows that ammonium nitrate explosives will usually detonate completely after storage for 6 months in a well-ventilated magazine.

Class 2, hydrated explosives. - To class 2 belong all explosives in which salts containing water of crystallization are the characteristic ingredients. The explosives of this class are somewhat similar in composition to the ordinary low-grade dynamites, except that one or more salts containing water of crystallization are added to reduce the flame temperature. They are easily detonated, and most of them can be used successfully in damp working places.

Class 3, organic nitrate explosives. - To class 3 belong all the explosives in which the characteristic ingredient is an organic nitrate other than nitroglycerin. The permissible explosives now listed under class 3 are nitrostarch explosives.

Class 4, nitroglycerin explosives. - To class 4 belong all the explosives in which the characteristic ingredient is nitroglycerin. These explosives contain free water or an excess of carbon, which is added to reduce the flame temperature. A few explosives of this class contain salts, or an unusually low percentage of nitroglycerin, that reduce the strength and shattering effect of the explosives on detonation. The nitroglycerin explosives have the advantage of detonating easily and of not being affected readily by moisture.

Class 5, ammonium perchlorate explosives. - To class 5 belong all explosives in which the characteristic ingredient is ammonium perchlorate.

Class 6, gelatin explosives. - To class 6 belong all explosives in which the nitroglycerin is gelatinized with nitrocotton.

Explosives of the last class have been grouped together at the end of the list because these explosives have been specially designed for blasting rock in coal mines, although under certain conditions they have been found suitable for shooting coal also. The rate of detonation given for these explosives is that determined when the explosives were submitted for test. It should be kept in mind, however, that this rate may range from 2,000 to 5,000 meters per second.

*** explosives enumerated on the permissible lists of the United States Bureau of Mines are permissible in use only when satisfying the following requirements:

1. That the explosive is in all respects similar to the sample submitted by the manufacturer for test.

2. That electric detonators are used of not less efficiency than those prescribed - namely, those consisting by weight of 80 parts of mercury fulminate and 20 parts of potassium chlorate (or their equivalents)- and that this electric firing must be done by means of a permissible-type blasting unit.

3. That the explosive, if frozen, shall be thoroughly thawed in a safe and suitable manner before use.

4. That the quantity used for a shot does not exceed 680 grams (1-1/2 pounds) and that it is properly confined with clay or other noncombustible stemming.

5. That the diameter of the cartridge used must be not less than that designated in the column "Smallest permissible diameter."

6. That the shot is not fired in the presence of a dangerous percentage of firedamp.

7. That the shot is not a dependent shot, is not bored into the solid, and does not have a burden so heavy that the shot obviously is liable to blow out.

An extended study has convinced the authors that, despite the favorable characteristics of permissible explosives as compared with black blasting powders and dynamites, which make them much safer in use, they are still not so foolproof if improperly used as to prevent accidents, including explosions due to the ignition of gas or fine bituminous-coal dust. However, permissible explosives are so devised that they produce a relatively small,

short-duration, low-temperature flame; they are therefore not nearly as likely to ignite inflammable mixtures containing explosive gas or coal dust as the dynamites or black blasting powders.

Permissible explosives are not so sensitive to sparks and flames as the black blasting powders or to friction or impact as most of the dynamites; moreover, they are not so insensitive (when properly handled) as to cause misfires.

Experience and experiment have amply proved that black blasting powders and dynamites are too hazardous to use in gassy mines. Permissibles are therefore extensively used in gassy mines or gassy portions of mines; their superior qualities commend them for use in all types of blasting in all coal mines, whether gassy or dusty or relatively free of both gas and dust.

Because permissible explosives are used extensively in gassy and dusty mines and generally are the only class of explosive used in very gassy mines, and thus are subjected to this hazardous condition of use, one might expect that many if not most of the ignitions of gas or dust from explosives would originate from permissible explosives. After diligent search the authors found only 13 instances of gas or dust ignition caused by or involving permissible explosives. In every one the explosive was definitely known to have been used in a nonpermissible manner - it was fired in a dangerous percentage of gas or adjacent to fine, inflammable coal dust, under one or more additional nonpermissible conditions of use. However, the Bureau has detailed records of 65 explosions involving black blasting powder only, 14 dynamite only, 10 black blasting powder and dynamite, and 11 black blasting powder and (or) dynamite with permissible explosives in the United States during the period 1908 to 1932, inclusive.

This circular is concerned mainly with the use of permissible explosives; hence but little attention is given the storage, underground transportation, or underground storage, except as they influence the quality of the explosive used and therefore are important in the proper use of permissible explosives.

Injuries to persons caused by accidents involving explosives are most frequently chargeable to improper use of the explosives; this statement also applies to permissible explosives. The greatest safety is attained when only those explosives having the greatest inherent factor of safety are used properly. Such explosives for coal mining are permissible explosives, since by certain pertinent tests permissible explosives have a factor of safety of at least 17 over common dynamite and at least 45 over the black blasting powders, on the basis of their liability to ignite gas and dust.

Permissible explosives are manufactured in a wide range of densities (or stick count), strength, rate of detonation, and diameters of cartridges (not less than 7/8 inch) and are classified on the basis of relative quantity of poisonous gases produced (class A the least poisonous gas, class B intermediate, and class C the greatest quantity of poisonous gas). Therefore,

there is a suitable permissible explosive for virtually any blasting problem in any coal mine. The low- to medium-density permissible explosives when skillfully used have proved as good lump-coal "getters" as any nonpermissible explosive. In recent years in Great Britain such low-density "permitted explosives" have supplanted Bobbinite - a compressed pellet powder - although Bobbinite had been used in gassy mines under special conditions for almost 30 years.4/ The authors hope that low-density permissible explosives likewise will supplant both the granular and pellet forms of black blasting powder as now used in many of the so-called nongassy coal mines of the United States.

The effect of an explosive upon the material to be blasted whether coal or rock, may be varied over wide limits by selecting the proper permissible explosive, by varying the number, position, and diameter of the boreholes, and by utilizing the proper diameter and length of the charge of the permissible explosive used. In other words, not only the character of the permissible explosive used, but also the method of using it, is important in this respect.

Where applicable, certain methods of using explosives are preferred; these will now be discussed.

PROPER PREPARATION OF FACE

The coal or coal and rock face should be prepared in a workmanlike manner; loose roof and coal should be pulled down, corners squared, and overhang eliminated as far as feasible. It is of utmost importance that an adequate number of properly directed holes should be drilled; much of the inefficiency and danger in blasting are due to drilling too few holes, correctly placed and spaced. If the face is properly trimmed and squared cracks or crevices in the face will be disclosed, and the holes can be drilled without intersecting these cracks or crevices. If feasible the coal should be cut or sheared, or both, and the sides of the cut or shear should parallel the surveyor's sight line; this will tend to prevent the drilling of holes on the solid, and blown-out shots will be reduced to a minimum.

The coal dust produced by cutting should be allayed by the use of water on the cutter bits and at the face; rock dust should be applied to within at least 40 feet of the face as a precaution against a dust explosion.

The cuttings or "bug dust" should be entirely removed from the cut or shear, thus permitting the coal to be blasted with less explosive; this precaution is important as regards both safety and efficiency. The bug dust should be loaded out before any hole is charged; the practice of throwing machine cuttings into the gob is hazardous and in the long run is likely to be expensive.

The gage of the drill or auger should always be large enough to drill a hole of adequate diameter, so that force is not required to charge the explosive; before the hole is charged all drillings should be removed from the borehole.

4/ Fifty-sixth Annual Report of His Majesty's Inspectors of Explosives, 1931, p. 5.

It is usually desirable to drill holes horizontally, but when it is desirable to throw the coal forward, as where mechanical loading is employed, the holes may be angled toward the right or left.

There should always be enough holes so that a reasonable charge of explosive will properly dislodge the coal or rock; this practice tends to produce a minimum quantity of slack, increase the percentage of lump coal, and prevent blown-out shots with their numerous hazards and inefficiencies. Although the proper use of explosive promotes safety and gives a maximum percentage of lump coal the explosive alone does not fully determine the percentage of lump; loading, transportation, and handling at the tipple also are controlling or at least contributing factors.

Holes should never be drilled back of or beyond the undercut or shear; they should usually end 3 to 6 inches in front of them and should be close enough to the rib to break it loose and close enough to the permanent roof to loosen the coal from the roof but not close enough to weaken, break, or shatter it. These results can usually be obtained by those skilled by experience in the art of blasting; no person who is not skilled in this art should be allowed to use explosives in any coal mine.

THE PRIMER AND ITS POSITION IN THE HOLE

Primers should be made with electric blasting caps, the cap secured centrally in the cartridge and pointing toward the bulk of the explosive in the primer. This is accomplished in different ways, depending upon the consistency of the explosive and the diameter of the cartridge. In the preparation of the primer the insulation of the wires to the electric blasting caps should not be damaged; otherwise the wires may be short-circuited and cause misfires. The wires should be brought off the end of the cartridge in which the detonator or cap is placed in such a way that when the primer is put into the hole last the detonator will point toward the bulk of the charge - that is, toward the bottom of the hole. The primer cartridge should be the last one to be charged, with the detonator pointing toward the bottom of the hole; experiments have indicated definitely that in the event of a blown-out shot the danger of igniting inflammable gas or dust is more remote than if the primer is placed anywhere else in the charge.

NUMBER AND PLACEMENT OF BOREHOLES

The number of holes required depends upon the width and thickness of the face to be blasted; whether the coal is undercut, centercut, overcut, or sheared or whether a combination of these methods of mechanical mining is used; and particularly upon the character of the bed and the nature and number and location of the bands of rock or other material. There should be enough holes to bring down the coal without using an excessive amount of explosive in any one hole; generally not more than 1-1/2 pounds of explosive should be used in any one hole.

The distance from the back of the hole to the nearest free face or machine cut should always be substantially less than, and should never exceed, the depth of the hole. Where the bed is thick or is tough or woody or where a strong or hard streak of rock, coal, or other material lies between the hole and the machine cut the holes should be arranged in two or more benches. In thin beds of coal that blasts easily narrow entries (12 feet or less) require at least 2 holes when the face is undercut, whereas wider places in the same coal probably should have at least 3 holes. In the Pittsburgh bed, where the coal is 6 to 7 feet thick and is both undercut and sheared, wide places (up to 25 feet in width) require at least 3 holes and narrow places 2 holes. If the coal is separated by a tough, center band or is thick and is machine-mined by undercutting it should be blasted in 2 benches at least, possibly 3 if the center band is particularly tough and requires separate blasting; such coal may require 1 center and 2 side holes in the lower bench and 2 corner holes in the top bench. Center-cut coal usually requires at least 2 holes in the top bench and 3 holes in the bottom bench for narrow work and an extra hole in both the upper and bottom benches for wide work. Safety considerations usually dictate that where coal is center-cut the top bench should be blasted first.

MACHINE MINING AS AN AID TO SAFE BLASTING

If coal is machine-mined, especially if both cut and sheared, small weights - 1/2 to 3/4 pound or 1 to 4 cartridges - of explosive per hole are often adequate, the number depending largely upon the size of the cartridge and the density of the explosive in the cartridge.

Where coal that blasts easily is both cut and sheared 1 pound of permissible explosive may blast as much as 20 tons of coal; where coal is shot off the solid and is woody and difficult to blast 1 pound of black blasting powder may produce less than one-half ton of coal, much of which may be in very fine sizes. Efficiency of explosives ranges between these extremes, being higher if coal is machine-mined, friable, free of rock or bony partings, and in a bed of moderate thickness.

Any increase in the efficiency of explosives aids safety because, in general, the smaller the quantity of explosive used per ton of product the smaller the hazard in its use.

CHARGING THE HOLE

If an adequate number of holes have been properly located and the shot firer has approached the face and assured himself that he is under safe roof and no gas is detectable he should proceed essentially as follows: If the hole has not been blown out with compressed air, scraped out, or otherwise cleaned he should remove the drillings with a scraper to prevent them from getting between the cartridges in the charge, leaving essentially no combustible material in the hole and preventing the cuttings from reducing the effective depth of the hole. He should then prove the depth, gage, and direction of the hole with his wooden tamping stick and measure the depth

of the undercut with the stick in the same vertical plane as the hole to be certain that the hole is not on the solid and hence beyond the limits of the cut. He should charge it by placing the cartridge or cartridges in the collar of the hole essentially in the following order: If there is a half cartridge or part of a cartridge he should place the cut end in the hole first, thus insuring that this end will be in the bottom of the hole. Moist or slightly moist explosive at the cut end is not likely to impede detonation of the balance of the charge. Then, if needed, one or more cartridges should be put in the collar of the hole and the entire charge, except the primer, pushed toward the bottom of the hole and gently seated there; the primer should now be made up in the standard manner and placed in the collar of the hole, making certain that the capsule is in the outer end of the primer and pointing toward the bottom of the hole; next, the primer should be pushed gently against the remainder of the charge with a wooden tamping stick, the wires being held so they will not be kinked or abraded while charging. The remainder of the hole should be filled to the collar with incombustible stemming material, preferably a mixture of molded sand and clay, although incombustible material in paper wrappers is suitable. If paper-container cartridges of incombustible stemming are used they should be made at least 14 inches long and each cartridge broken in the middle; the broken end should be placed in the hole first to make sure that there is little or no paper in contact with the explosive. Furthermore, the weakened forward end of the cartridge will therefore upset when rammed and facilitate filling the borehole with stemming material.

Permissible explosives usually should be charged without breaking the cartridges and seldom require vigorous ramming or tamping. If a charge of high density is required the cartridges, other than the primer, should be slit and pushed firmly into place one at a time but not violently.

In placing the stemming the first portion of it should be pressed firmly or gently in place as required, but the last portion may be tamped more vigorously.

An important precaution to reduce the hazard in charging the borehole is to protect the explosive and the electric blasting cap used for firing the explosive from all impact, friction, sparks, and flames, as these are the most frequent causes of premature blasts in charging.

If permissible explosives are used instead of black blasting powder, especially black blasting powder in granular form, there is relatively little danger of ignition by sparks, flames, and their sources; this is one of the main reasons why black blasting powder should be supplanted by permissible explosives in coal mines.

WHO SHOULD DRILL HOLE, CHARGE, TAMP, AND FIRE?

In these days of mechanical mining it is feasible in many mines to drill holes with either compressed-air or electric drills, but whether they are drilled manually or with power the drilling should be done by a special crew

upon whom dependence can be placed to locate the holes in accordance with a standard adopted for each type of working face. Where holes are drilled by individual miners it generally is useless to expect correct location of holes; improperly located holes place upon the shot firer the disagreeable task of refusing to charge or fire them or taking a chance of losing his life if he should fire them. Many shot firers undoubtedly have been maimed or killed through holes carelessly or even maliciously loaded by persons not required to fire them.

Competent shot firers should charge and tamp holes, and their competence should be attested by a State certificate as shot firer, mine examiner, fire-boss, or foreman; as a reasonably safe alternative for actual loading by a certified man the shots may be charged and the holes tamped under the immediate supervision of a man so qualified.

Decision 12 of the Mine Safety Board of the United States Bureau of Mines covers this point:

> 5. Each shot employing explosives shall be prepared and fired by or under the immediate supervision of a man having a State certificate as a mine examiner, fireboss, or foreman; and whenever conditions permit all other men than those authorized to prepare and fire shots shall be out of the mine when shot-firing with explosives is being done.

Next to firing from outside the mine when no one is in the mine the greatest safety is assured if shots are fired on the "off" shift when no one is in the mine except the shot firer; in the handling and use of explosives it is a fundamental fact that hazards to life are least when the fewest number of persons are in the danger zone. This method of firing shots prevents the fatalities and nonfatal injuries classified as gas- and dust-explosions, injuries caused by explosives, unguarded shots, returning too soon, delayed blast, hit by projected material of blast, and shot breaking through rib or pillar.

TESTING FOR GAS

Both before and after charging a hole the shot firer should assure himself by a suitable test, such as with a permissible flame safety lamp, that no gas is present; if he finds gas before charging in quantities that indicate danger the hole should not be charged, and if he finds it after the hole is charged the shot should not be fired until after the gas has been removed by ventilation. Although permissible explosives under test conditions stand tests indicating that they do not ignite methane, there are numerous conditions in mine workings under which gas or dust may be ignited even when permissible explosives are used; the use of permissible explosives should not be relied upon to counterbalance defects in ventilation.

The time from testing for gas to the instant of firing should be as short as possible, and the intervening duties should be expedited.

REMOVING ELECTRICAL SHORT CIRCUIT AND CONNECTING TO SHOT-FIRING CABLE

When shots are fired by electricity the electrical short circuit on the legs of the electric blasting cap should be removed immediately before the shot-firing cable is connected to it. The shot-firing cable should be tested to make sure that it carries no current, and the uninsulated portion of the legs of the electric blasting cap should be connected to the shot-firing cable by an efficient joint.

To prevent commotion at the face when a shot is fired (caused by movement of coal or roof or escape of gases) from bringing two uninsulated portions of the shot-firing circuit together and precipitating a "break spark", the joints should be a few feet from the collar of the hole and arranged as far apart as feasible; they also should be taped and staggered to prevent their contact, but fairly good protection is offered if they are staggered and separated as far as possible even though they are not taped. Tools should be removed before this final face connection is made and the cable unreeled promptly afterward.

UNREELING THE CABLE AND SEEKING A SAFE PLACE

The cable should be at least 100 feet long and should be kept dry and in good repair. In unreeling the cable toward the point of firing care should be taken that it touches no rails, pipe lines, electric wires, or electric machines; the best way is to hang the cable on insulators, which may be dry wooden pegs or cappieces or props.

After the shot firer has reached the point selected for firing, which is known to have a safe top and which is definitely around a corner or not in a straight line from the point of firing so that projected material from the blast will not be thrown into it, the shot may be fired. The cable should be at least 100 feet long to permit the shot firer to fire from a safe place. The battery end of the cable should be "shorted" at all times except while the shot firer has assured himself of being in a safe place and is actually firing.

GUARDING, WARNING, AND FIRING

If no other men are in the mine and the firer is sure that the men with him are in a safe place (he should be the nearest one to the shot) he should call in a loud voice "Fire", remove the "short" from the battery end of the shot-firing cable, attach the cable to the shot-firing unit, loudly call "Fire" again, and then fire within a second or two.

If other men are in the mine all in nearby working places should be warned, and all crosscuts and other approaches to the place should be guarded; if the blast is in a face approaching other workings those in the other workings also should be warned before the shot is fired. The shot firer should then call "Fire" loudly, remove the "short" from the end of

the shot-firing cable, attach the cable to the blasting unit, call "Fire" again in a loud voice, and then blast within a second or two. Immediately after a charge is fired the shot-firing unit should be detached from the firing cable and the firing cable "shorted."

METHODS OF FIRING SHOTS

The type of accident that can be avoided by firing shots from outside the mine has already been indicated; this method of firing shots from outside the mine when no one is in the mine is greatly to be preferred and is practiced successfully in several coal mines in Utah, Colorado, New Mexico, and elsewhere. As with all methods, most if not all of the hazards in transportation, charging, drilling into old holes, and premature shots still exist, though they can be rigidly safeguarded; the safety records of mines using this system are remarkably good as regards blasting accidents of all kinds, including the occurrence of fires and explosions; and injuries or fatalities to workers are almost eliminated.

Other methods of firing, in the order of their preference, are:

1. Charging and firing by shot firers when all other men are out of the mine.

2. Charging and firing by shot firers on the "off" shift.

3. Charging and firing by shot firers on the regular shift.

4. Charging by miners during the shift and firing by shot firers when all other men are out of the mine.

5. Charging and firing by miners at the end of the shift or at some specified time during the shift, such as midday.

The method least to be preferred is:

6. Charging and firing by miners at any time during the shift.

Charging and firing by shot firers are in all cases preferred, as there are many advantages in all blasting operations being handled by certified or qualified shot firers; men with such qualifications are competent and are given a reasonable amount of sensible instruction and supervision. It is important that they be given such instructions and supervision; otherwise human frailties will intervene, and recklessness of various kinds creep into the work. A shot firer should examine the place for explosive gas and dangerous roof before and after shooting and should see that enough holes have been drilled and that they are properly placed. He should be relied upon to use the right amount of explosive and see that it is correctly charged and tamped. He also should be expected to adhere to safety standards, such as warning men in adjacent working places, if blasting is being done while men are in the mine, and guarding approaches to the place. Well-selected shot firers

are more likely to fire from a safe distance than the miners and if they are carefully supervised should make sure that coal is blasted in an efficient manner, usually with a larger production of lump coal, reduction in explosive used, and minimum harm to timbers, roof, or other mining adjuncts.

Method of Firing Shots in Utah

A detailed description of the method of firing from outside the mine, as practiced in a mine in Utah,[5] follows:

All shots are fired electrically from outside the mine with no one underground, using 220 volts alternating current. No. 6 wire is used on the blasting circuit, except in rooms, where No. 14 wire is used. A switch in the circuit on each entry near the slope is left open when not in use. In addition, there are 3 switches in the shot-firing circuit near the manway portal; 2 of these are double-pole knife switches, and 1 is operated by a pipe gate. The switches are locked open until all shot firers are out of the mine. After the 2 knife switches are closed the gate switch is closed and locked. The circuit is then complete between the mine and the master or timing switch in the nearby check cabin.

Each of the 3 shot firers has a key to only 1 of the switches near the manway portal; all of the shot firers must therefore be out of the mine before blasting. In other words, the circuit cannot be completed in the mine until all the shot firers reach the surface, unlock their respective switches, and close them.

After the three switches have been closed the time-limit switch is ready to be operated. It is first unlocked and the plunger, which operates in a slotted 1-inch pipe, is raised and connected to the solenoid switch. The current flows through the firing circuit during the time that the plunger drops 6 feet 8 inches, which is about 0.7 second. The electric blasting caps are sometimes connected in parallel and other times in parallel-series.

Detailed information on methods of firing in other Utah mines where firing is done from outside the mine when no one is in it follows. In one mine the coal is undercut in 25-foot rooms and blasted in 2 benches - 4 holes in the bottom bench and 5 holes in the top bench. The 4 holes in the bottom bench are connecced in series, the 5 holes in the top bench are connected in another series, and the 2 series are connected in parallel. The 2 center holes of the 4 lower holes are fired by instantaneous electric blasting caps, and the 2 side holes have first-delay electric blasting caps; the center hole of the upper 5 holes has first-delay, the 2 intermediate holes have second-delay, and the 2 side holes have third-delay electric blasting caps.

5/ Parker, D. J., Safety Practices at Mine 1, Spring Canyon Coal Co., Utah: Inf. Circ. 6675, Bureau of Mines, 1932, 10 pp.

In another mine where the coal is undercut 4 holes connected in parallel are sometimes used for blasting in 1 bench. At other working places in the same mine coal is blasted in 2 benches - 4 holes for the bottom bench and 4 holes for the top bench - and the holes are connected in parallel-series with 2 electric blasting caps in each series. In another mine coal is undercut and blasted in 2 benches with 4 holes in each bench; all holes are connected in parallel.

Live current is used for such blasting; the voltage should always be adequate so that when any holes are connected in series or parallel-series at least 1.5 amperes are available for each electric blasting cap, and when parallel connections are used at least 1 ampere is available for each electric blasting cap. The voltage required varies with the resistance in the shot-firing circuit, and this depends upon the gage of copper wire used and the distance between the firing switch and the working faces. To prevent misfires a current not less than that specified above must be provided. It is well to keep the voltage as low as is feasible to prevent loss of current through leakage along the shot-firing circuit. Power need not be applied in the shot-firing circuit as long as 0.7 second (700 milliseconds, a millisecond being 1/1000 of a second) even when electric blasting caps are connected in series, because 5 milliseconds have proved adequate in tests where the source of power was a powerful, manually operated shot-firing unit of the generator type. With a controlled current value certain types of instantaneous electric blasting caps do not fire in less than 7 milliseconds. The combination of a 5-millisecond timing switch in the shot-firing circuit at the source of power with this particular type of electric blasting cap insures that there is no voltage (or current) on the shot-firing circuit at the instant the shots explode. This reduces substantially the opportunity for ignition of inflammable gas or dust mixtures by a break spark on the firing circuit caused by commotion at the face due to the blast.

RETURN TO FACE AFTER BLASTING

After a blast it is unwise to return to the face immediately, even though the shots appear to have fired properly, because occasionally a single shot may explode twice. The reason for this double fire is obscure, but it has been attributed to insensitive or deteriorated explosive, usually caused by improper storage; only a portion of the charge explodes at first, and later another portion is ignited by the heat or burning of the first explosion. Weak electric blasting "caps" may cause a somewhat similar result; these caps should be of No. 6 grade or stronger. No cap of less than No. 6 grade is manufactured in the United States for use in this country. Possibly in these double-fire cases the charge did actually explode completely, but the material did not yield to the high pressure within the borehole until later, and this delayed yielding sounded like a second explosion. There are numerous other reasons why one should not return to the face immediately after shot-firing. The smoke and dust may so obscure the roof or face that immediate approach is hazardous, the partly loosened material may not have finished falling, or the explosives fumes and dust may not have been adequately diluted by ventilation, and noxious gases, fumes and dusts may be

present. Dangerous gases are least likely to be encountered if a permissible explosive of class A is used and if the weight of the charge is small (preferably less than 1-1/2 pounds per hole).

MISFIRES AND HANGFIRES

Prevention of Misfires

Every effort should be made to prevent misfires. Those known to have occurred must be handled almost invariably, and handling these is always hazardous. Sometimes, however, misfires, especially partial misfires, are not detected at the time of firing but may be encountered by drilling into them possibly months or even years later. Whether the misfire is known or unknown, the explosive may be hidden in the coal or rock and remain a continual menace until it is either discovered and removed or until it is burned with the coal, with or without an explosion.

Misfires can be prevented by selecting proper explosives and keeping electric blasting caps in first-class condition by proper storage; using the explosive before it has aged unduly; and correct storing, priming, charging, and firing of the explosive. The officials of each mine and of each mining company should maintain a detailed record of the number of holes fired on every shift, number of misfires, and number of accidents in handling misfires. Such a record would show type of explosive, make of electric blasting cap, type of primer, and method of firing most likely to produce misfires; number of accidents chargeable to each method of handling misfires; and the miner or shot firer responsible for misfires. These data should aid in gradually evolving the best type of explosive and electric blasting cap for use in the particular mine, the safest method of firing, the best type of primer, and the safest method of handling misfires for each condition encountered. The publication of such details by the management would contribute materially to the prevention of misfires in mining.

Careful electric shot-firing offers the best means of reducing the number of misfires, but electric firing is by no means foolproof and should not be inaugurated without proper education and instruction of mine workers, especially supervisors and shot firers. Avoidance of misfires in electric firing presupposes the use of a good type of explosive in good condition, use of well-prepared primers, skill in charging and tamping the holes, a good firing circuit, and a blasting unit of adequate capacity.

Misfires in electric firing may result from conditions outside the borehole, such as poor methods of storing explosives, damp or deteriorated explosives, open circuit, short circuit, or too much resistance in firing line, or not enough capacity at source of electric current. These possible defects should be investigated first, especially the electrical defects; for this purpose a circuit tester is useful. Care in making connections, inspection and proper maintenance of shot-firing cables, and periodic testing of the blasting unit to make sure that it is up to capacity will usually

prevent misfires from this source. When the shot firer is sure that the cause is within the borehole the misfire must be handled, preferably by the shot firer, by no means a desirable duty.

Return to a Misfire

As soon as one or more attempts have been made to fire a shot electrically the blasting unit should be disconnected promptly from the shot-firing cable and the end of the cable short-circuited.

The time interval before return to the face provided for in most State laws or company rules should be observed, but in the absence of such laws or rules 15 minutes at least should elapse before return; when conditions permit, this minimum should be greatly increased not only because of the possible shot-firing hazards but also because of possible loose roof or coal or dust and gases in the air if other holes had fired.

Any misfire may prove to be a hangfire or a delayed blast; hence a reasonable time should elapse before return to a misfire. The belief that at least 8 hours should constitute the "reasonable time" is warranted.

Handling Misfires

State regulatory agencies may prescribe the method of handling misfires. Such rules usually state that in coal mines a new hole shall be drilled alongside the misfired hole and approximately 1 foot from it, so started and slanted that there will be no chance for the new hole to intersect the misfired hole; the new hole should be charged lightly and blasted in the usual manner. Company rules are sometimes more rigid than State requirements and, of course, should in no case be less rigid. The object of the blast in the new hole is to expose the charge in the misfired hole so that it can be recovered completely; the charge in the second hole, therefore, should be light to avoid exploding the misfired hole. The second hole should be on the free side of the misfired hole; the direction of the hole is more important than the distance from the misfired hole, as it is important that the second hole should not be drilled into the first hole. The primer of the first hole can be recovered more readily if the wires of the electric blasting cap of the misfired hole are anchored outside the hole.

Misfires should be handled only by competent persons (shot firers or supervisory officials) and, where at all feasible, under the direction or immediate supervision of the foreman or assistant foreman. They should be handled preferably on an off shift, and until they are recovered no person or persons should be allowed to work in the working place in which they are located.

Where regulations permit and water is available at the face stemming in misfired holes may be removed carefully with water under pressure applied at the hole through a rubber hose; the pressure, however, should not

be excessive. This method is facilitated and made less hazardous by the general use of sand or a mixture of sand and clay for stemming and by special incombustible indicating materials in the stemming next to the explosive, such as iron oxide; or if wrappers are used on the stemming a distinctive color of stemming in the first cartridge of stemming may be placed in the hole or a distinctive color used for the paper of that cartridge. When the stemming is washed out and the cartridge exposed a new primer may be inserted, the hole restemmed, and an attempt made to fire the charge in the usual way. This work should be done without undue delay; otherwise the water may so desensitize the front of the explosive charge that it will not explode completely, if at all. The stemming of the explosive should never be removed with a drill, auger, or other metal tool, but when the stemming is being removed by water the operation may be facilitated by the use of a long wooden spoon. Neither the charge of explosive nor any portion of it should be removed from the hole; pulling a charge from a misfired hole by the lead wires is especially dangerous.

Misfires should occur no more frequently than 1 for every 5,000 shots, or 0.02 percent of the number of shots fired. If they occur more frequently something is wrong with the explosive, the electric blasting cap, the method of preparing the primer, the method of charging the hole, the firing circuit, or the blasting unit, and an investigation should be made to determine just where the trouble lies; possibly the entire blasting system is at fault because it is not applicable to the conditions at hand.

NOXIOUS-GAS HAZARD FROM EXPLOSIVES

Permissible explosives when detonated give off gaseous products of combustion, a small to moderate percentage of which is carbon monoxide, depending upon the class of explosive used. Most permissible explosives, even when properly and completely detonated in a drill hole in a coal mine, produce the poisonous gas carbon monoxide and a much larger volume of non-poisonous gases. Permissible explosives, however, give off much less dangerous gas than other explosives used in mining and quarrying. In order that the poisonous gases may not become a menace to the lives or health of miners under normal conditions, no explosive can become permissible if upon detonation of 1-1/2 pounds it evolves more than 158 liters (about 5-1/2 cubic feet) of poisonous gases, as determined by tests in the Bichel pressure gage. The classification of permissible explosives on the basis of the volume of poisonous gases produced by 680 grams (1-1/2 pounds) of the explosive is as follows: Class A explosives, from which the volume of poisonous gas produced is not more than 53 liters; class B explosives, from which the volume of poisonous gas is more than 53 but less than 106 liters, inclusive; and class C explosives, from which the volume of poisonous gas is more than 106 but less than 158 liters, inclusive.

Where ventilation is sluggish, miners or shot firers should not return to the face for at least 5 and preferably as much as 15 minutes after a shot, even when permissible explosives have been used; at all working faces with little or no circulation of air explosives of class A or B should be used,

preferably class A. The quantity of noxious gas produced is directly proportional to the quantity of explosive used; therefore the efficient use of explosives is highly desirable from this standpoint alone.

Irrespective of the efficiency of ventilation, however, it is difficult to prevent poisonous gases from remaining in the voids in a pile of blasted coal or rock; ventilation of these voids to drive out the noxious gases has in some cases been promoted by sprinkling the pile with water. When conditions permit it is well to blast the coal some hours before it is loaded - one of numerous reasons for firing shots between the regular shifts.

Compared with other explosives the least poisonous gas is produced by firing gelatin dynamites, ammonia dynamites, and class A permissible explosives, but the unsuitability of gelatin and ammonia dynamites in other respects does not commend them for use in coal mines.

Black blasting powder, in any or all forms, is likely to produce considerable carbon monoxide and some hydrogen sulphide; the hydrogen sulphide, however, probably does not persist for long, as it has a tendency to "break down", especially under moist or wet conditions. The black blasting powder known as "pellet powder" is by no means free from many of the hazards associated with granular black blasting powder; numerous cases of suffocation or sickness from breathing the fumes after blasts of this explosive have been reported. Unquestionably, some of the trouble is due to the widespread opinion among users and proponents of this explosive that the noxious gases produced are not hazardous; hence men return too soon to the face after blasting. The quantity of noxious gases produced by this explosive is probably enhanced by the paraffin-paper wrapper; improvements in pellet powder are claimed to have resulted from either a reduction in the quantity of paraffin on the wrapper or a change in the quality of the paraffin used. No general statement regarding these features is warranted at this time, except that apparently there is considerable difference in the quantity and composition of gases produced from pellet powder made by different manufacturers or from different grades of pellet powder made by the same manufacturer or from any grade of pellet powder under certain conditions of use. In the near future some competent agency should remove this question of noxious gases produced by pellet powder from the field of opinion, and possibly prejudice, into the field of certainty by making adequate tests under operating conditions. At present pellet powder lacks much of being a desirable explosive for use in underground operations, not only as to irrespirable gases evolved but also as to the hazard of starting fires or explosions by ignitions of gas or dust or both.

Straight nitroglycerin dynamite should never be used underground or in confined or poorly ventilated places; the high percentage of carbon monoxide produced renders it decidedly unfit for such uses. Numerous deaths have occurred where this type of explosive has been used in mines and to some extent in quarries.

SMOKE PRODUCED BY EXPLOSIVES

The smoke produced by permissible and other explosives depends in part upon the brand of explosive used but largely upon the conditions of use, including storage. If the shot is confined properly in a borehole and fired with a suitable primer a minimum of smoke should be produced. Smoke, in addition to being disagreeable and possibly hazardous to breathe, reduces visibility. Therefore only permissible explosives, which in practice produce the least smoke, should be used provided they are suitable otherwise; moreover the workers should not return to the face immediately after a shot is fired, and ample ventilation currents should be provided to move or at least dilute smoke promptly and efficiently.

POSSIBILITY OF DEVISING BETTER PERMISSIBLE EXPLOSIVES

The question frequently has been asked as to the possibility of devising a permissible explosive that will ignite neither gas nor dust, regardless of the conditions under which it is used; the characteristics of explosives are such that as yet no such explosive has been devised or is likely to be devised in the immediate future. Sensible mining men therefore will use due caution in the firing of any kind of shots at any time, as there is no such article as a perfectly safe explosive.

The energy of a permissible explosive, like that of other explosives, results from the almost instantaneous production of a very large quantity of highly heated gases which, when properly confined, provide the pressure that brings down the coal or rock; the high pressure of the gas is due chiefly to the heat of the gases, and reduction in temperature is almost sure to reduce the strength of the explosive correspondingly. Hence, any reduction in strength would be balanced by an increase in the quantity of explosive used per hole, and any gain in safety through reduction in temperature would be lost in large part because of the larger quantity used. More important, however, is the fact that numerous efforts to improve the safety and working efficiency of permissible explosives to the point where ignition of gas is impossible have resulted in types of explosives so insensitive to detonation that misfires and partial misfires occurred frequently and the explosive was unsuitable from the standpoint of sensitiveness to detonation. Until substantial assistance can be expected in this direction, how can maximum safety in the use of permissible explosives be obtained? Obviously, at present about the only recourse is to use suitable safeguards against the explosives. Several of these, which can be applied easily, are set forth in detail in this publication; adequate instruction, supervision, and discipline in their use makes them effective.

SPECIAL APPLICATION OF SHEATHED EXPLOSIVES

An unusual hazard obtains where a crevice intersects the borehole and a free face. Sometimes these crevices exist before the hole is drilled; sometimes the crack is opened by firing nearby holes; and sometimes in friable coal which is undercut a portion of the coal drops and opens a crevice, even

after the hole is charged, if it is not fired promptly. Some such crevices may be observed easily, and others are difficult to detect; the holes in which crevices are found should be inspected carefully, if necessary using a special tool such as a rod having a sharp side projection or a mirror. The hazard arises from the fact that upon detonation of a charge the gases escape forcibly and readily into the atmosphere through the crevice and are likely to ignite inflammable gas or dust mixtures if they are present. A measure of protection against this hazard is afforded by enclosing cartridges of permissible explosives in a sheath of special-quality sodium bicarbonate, as is done in Great Britain, or of calcium fluoride, sodium chloride, and plaster, as is done in Belgium; the sheath ordinarily is one-eighth inch thick, and the "sheathing" is done at explosives factories.

Explosives so protected are larger in diameter, heavier, and cost more than unsheathed explosives. Limited tests show the sheathed explosives to be little less efficient than unsheathed explosives but much less likely to ignite gas or dust when the explosive is used in relatively large quantities per hole.

Tests abroad have shown that the "limit charge" of sheathed permitted explosives is 2 to 3 times as much as that of unsheathed permitted explosives. In Belgium sheathed explosives represent more than 30 percent of all explosives used to bring down coal. Sheathed explosives would undoubtedly be much safer than ordinary explosives where conditions demand or are thought to demand large charges (more than 1-1/2 pounds per hole); they also would reduce the hazard if used for blasting in chutes in pitching coal beds where the accompanying hazards do not permit the drilling of holes or the proper confinement of a charge in a hole.

Users of explosives should keep sheathed explosives available for these special applications; it is not known whether explosives manufacturers in the United States make sheathed explosives for these purposes, but undoubtedly they would make them if a real demand should arise. The experience of Great Britain and Belgium with sheathed explosives appears to indicate that safety could be increased in connection with some abnormal types of blasting in the coal mines of the United States if sheathed explosives were used.

CONDITIONS UNDER WHICH PERMISSIBLE EXPLOSIVES SHOULD BE USED

Permissible Explosives Should Not Be Fired in a Dangerous Percentage of Firedamp

Decision 12 of the Mine Safety Board of the Bureau of Mines states:

3. Before and following each shot in gassy and slightly gassy coal mines examination for gas shall be made with a permissible flame safety lamp or permissible equivalent, and

4. If more than 1-1/2 percent of inflammable gas is found, in the quantity and by the method specified in Mine Safety Decision 9, the place shall be considered to be in a hazardous condition and before another shot is fired the gas shall be reduced by ventilation below the percentage and quantity specified in Decision 9.

Decision 9 of the Mine Safety Board of the Bureau of Mines states:

5. If the air in any unsealed place, when sampled or tested in any part of that place not nearer than 4 feet from the face and 10 inches from the roof, shall be found to contain -

(a) More than 1-1/2 percent of inflammable gas, the place shall be considered to be in hazardous condition and require improved ventilation, and

(b) If more than 2-1/2 percent of inflammable gas is found, the place shall be considered dangerous, and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said place.

If control of ventilation were always so perfect that there would never be any inflammable gas in a place in which blasting is done the hazards of gas or dust ignition with blasting would be confined to the ignition of inflammable mixtures of dust and air, in itself no negligible hazard. Experience has amply demonstrated that such conditions are not attained in practice; far too frequently a dangerous percentage of gas or of gas and dust is present when shots are fired. After a gas ignition or a gas explosion it has frequently been found that, although the place was tested and found clear of gas upon completion of charging the hole, during the interval between testing and firing the shot the ventilation was deranged by a fall, an open door, or otherwise, and at the instant the shot was fired explosive gas was present; if several shots are fired one after another either with or without the use of delay detonators explosive gas or dust or both are likely to be present after the first shot is fired. Likewise, it is not uncommon to find after a gas ignition or gas explosion either that no test for gas has been made or that testing has not been done properly; moreover blasting has been done when explosive gas was definitely known to be present, many persons through ignorance or otherwise placing undue confidence on the safety features of permissible explosives.

The blasting operation should be conducted so that the ignition of gas will be as unlikely as is humanly possible - the place to be blasted should be adequately ventilated and carefully and properly tested for gas immediately before the shot is fired.

<u>Rock-dusting to guard against ignition of inflammable coal dust</u>. - All coal dust, except possibly that from some grades of anthracite, is definitely inflammable; this fact should at all times be recognized, and dust ignition and propagation by gas explosion or directly by explosives should be prevented

The Bureau has a record of a "bulldozing" shot which threw dust into suspension to be ignited by a live current used to fire the shot of a permissible-branded explosive. The explosive was not in a borehole and was therefore not confined properly; hence it was not used in accordance with the permissibility requirements of the Bureau of Mines. Blockholing should have been done and the shot fired with a permissible shot-firing unit which provides an electric current not potent enough to ignite gas or dust. The Bureau has a record of two other gas and dust explosions in which permissible explosives were used; it was uncertain whether gas or dust was ignited (either may have been) and whether the ignition was caused by a blown-out shot or the electric current used for firing. The propagation of the explosion by dust could have been eliminated or greatly restricted if rock-dusting had been adequate (see Bureau of Mines Report of Investigations 2606 and Information Circular 6030). Rock-dusting the rooms and scattering rock dust in the vicinity of shots, in addition to rock-dusting all entries, slopes, and passageways, probably would have prevented much of the damage in these instances. Rock-dusting is very useful if shots are improperly confined in boreholes, if holes are not correctly placed, or inflammable gas is present, but it certainly should not be used as a "cloak" for dangerous practices in the use of explosives or of dangerous types of explosives to obtain less slack or more lump coal.

Proper Confinement of Charge in Drill Hole

In accordance with Decision 12, item 1, of the Mine Safety Board of the Bureau of Mines -

1. Each charge shall be in a hole properly drilled and stemmed with incombustible material.

The active list of permissible explosives as of August 1934 (Report of Investigations 3259) stipulates -

4. That the quantity used for a shot does not exceed 680 grams (1-1/2 pounds) and that it is properly confined with clay or other incombustible stemming.

In other words, the charge of explosive must be so isolated from inflammable gas or inflammable dust mixtures that the shot will not ignite these mixtures.

It has long been recognized that one of the most hazardous conditions that attend blasting is a blown-out or "windy" shot, and this is likely to occur frequently when blasting off the solid. As shots in blowing out commonly tear away the collar of the hole and eject the stemming from it, this type of shot is considered exceptionally hazardous. Unquestionably it is hazardous; so also are other types of blown-out shots.

Shots that blow out through the collar of the hole may be prevented by blasting with an adequate number of holes and by not requiring the explosive to handle an excessive amount of burden; one way of accomplishing this is to

undercut or shear the coal where at all feasible and to use enough of the most effective and efficient incombustible stemming material. The Bureau of Mines advocates that clay, sand, rock dust, or combinations of them fill the requirements because they are incombustible stemming materials. Recent tests in Great Britain have indicated that a 3:1 mixture of moist sand and clay "used neat" (that is, without a wrapper and prepared as "dummies") is not only an efficient but also a practicable and readily available type of incombustible stemming. If blown-out shots could be prevented manifestly there would be no ignitions of gas or dust from this cause, regardless of the kind or type of stemming used.

Under some conditions shots may also blow out the back of the hole, ignite inflammable gas or dust, and fatally injure persons in the vicinity. As a precaution against this type of blown-out shot the back of all holes which approach open working places - this is most frequent in "holing through" crosscuts, rooms, or headings to adjacent workings - should as far as feasible be drilled so that as much stemming can be placed in the back of the hole as in the front; that is, such holes should be considered as having two ends, each of which must be stemmed. In fact, when holing through a hole frequently does have two open ends.

In another type of blown-out shot the charge is placed so close to a free face that when the shot is fired it blows out sidewise, downward, or upward because of the inadequate burden in that particular direction, with the possibility of igniting inflammable mixtures of gas or dust. If the bottom of the shot hole is placed too close to an undercut, overcut, or shear or if there is a crack or crevice or open cleat from the end or approximately the end of the hole to the undercut, overcut, or shear the shot may blow through with disastrous results.

This condition is likely to occur if dependent shots or possibly those using delay detonators with two or more delays to the same face are fired, because a previous shot has so dislodged, removed, or loosened what should have been the normal burden for the succeeding shot that the succeeding shot may blow out. On occasion this burden may be removed to the extent that the succeeding shot is actually fired in the open. Moreover, a blown-out shot is likely to occur when two or more holes in the same working place are charged before the first hole is fired, even though each hole is fired separately. The best precaution to prevent a blown-out shot due to an adjacent shot is to charge and fire the first hole before the second hole is charged, to charge and fire the second hole before the third hole is charged, and so on until all shots in the place are fired. In this way no improperly placed hole need be charged, and there will be no charged hole requiring a choice to be made between handling it as a misfired hole or firing it. Sometimes the safest procedure is to fire the several holes simultaneously.

First-Class Condition of Explosive Required

One of the permissible conditions of use of a permissible explosive is "that the explosive is in all respects similar to the sample submitted by the manufacturer for tests."

Every explosive submitted to the Bureau of Mines for permissibility tests and placed on the active permissible list has certain basic chemical and physical characteristics. Chemically the components should be of the same kind and proportions and mixed in the same sequence and manner as the explosive on which the Bureau permissibility tests were made. Physically the explosive must for each size of cartridge be within certain tolerances of weight and character of wrapper; have the same strength (unit deflective charge), rate of detonation, and poisonous gases; and be sensitive enough to detonation and not too sensitive to frictional impact. These attributes are a prerequisite and an assurance of the continuing permissibility of each brand of permissible explosive. Each permissible explosive is submitted by the manufacturer and tested by the Bureau of Mines and may be checked from time to time by the Bureau, which on its own initiative may procure field samples of permissible explosives and check them against the various chemical and physical properties of the explosive as found in Bureau of Mines permissibility tests. The manufacturers have earned a high reputation in this respect. However, the matter of permissibility of an explosive does not end here; the condition of explosive when used may differ materially from that when received from the manufacturer, as the condition of the explosive depends largely upon the manner in which it is stored. Experience has shown that substandard storage conditions injurious to explosives are the rule rather than the exception. Explosives, particularly permissible explosives, should be considered perishable commodities, as extremes of heat, high moisture, or long-time storage affect them adversely. All permissible explosives of the ammonium nitrate class absorb moisture readily in moist, damp, or wet places if the paper wrappers are broken and may become insensitive if stored thus for long periods, especially at temperatures of 90° F. or higher. The absorption of moisture by the ingredients of the cartridge may in a very few days cause a portion of a broken cartridge to become so insensitive that complete detonation of the charge cannot be assured; this often results in misfires, partial misfires, or the burning of a portion of the charge or the entire charge rather than effective detonation.

The operator or user should therefore: (1) Buy explosives in such quantities that they may be used promptly; (2) store the explosives in main storage magazines and distributing magazines so that the oldest stock of any brand may be used first; (3) use the older stock first; (4) transport and handle the explosives so that the wrappers are not broken, preferably in rugged containers such as boxes; (5) subject explosives to moist storage, especially underground, the shortest possible length of time - only the quantity of explosive estimated to be needed for use that day should be taken underground, and any left-over explosive should be returned to the better surface storage at the end of the shift; and (6), if portions of cartridges are used in a hole, use the remaining portion as promptly as possible and when charged in a hole place the cut end of this portion of the cartridge at the bottom of the hole.

Prevention of Mine Fires Caused by Explosives

Numerous coal-mine fires have been caused by explosives or by the means used to fire the explosives. Such fires were particularly numerous many years ago when granular black blasting powder was used extensively and often improperly. Fleming and Koster[6] state:

> Furthermore, every mine in the district (Franklin County, Ill.) suffered from frequent fires from the use of black blasting powder, the coal igniting readily and there being some methane liberated at the face. At some mines as many as 25 fires occurred nightly, following the firing of shots, and in two instances 40 were reported. The employment of fire runners to follow the shot firers and extinguish fires became necessary. Four to 10 men were employed for this work at each of the various mines, and the relatively small number of fires getting beyond control speaks well of the efficiency and carefulness of these men and of the officials.

Fires caused directly by black blasting powder or by the fuse or squibs used to fire it and fires following mine explosions caused similarly were very expensive (see Bull. 137, p. 9):

> *** a heavy expense must be charged against explosions and fires from the use of black blasting powder. *** an explosion occurred at shot-firing time, which killed the shot firers and set the mine on fire ***. Both shafts had to be sealed. The shafts were kept sealed for 120 days, and later the mine was flooded ***. The total loss due to flooding, suspension, and restoring the mine to working conditions was upward of $175,000.

The operators and the miners agreed upon an improvement in the method of using black blasting powder, but this gave only partial relief from mine fires, which continued to occur at shot-firing time, even when black blasting powder was used by expert miners under stringent regulations. The introduction of permissible explosives resulted in virtual elimination of mine fires caused during the shot-firing operation. However, some fires did occur with permissible explosives because of the use of excessive charges, little or no stemming, deteriorated (damp) explosives, mixed charges in the same borehole, or fuse and "cap" instead of the electric blasting cap; in other words, the explosives were used in a nonpermissible manner.

Granular black blasting powders continue to cause mine fires. In 1928 a mine in Illinois averaged 3 fires a week from granular black blasting powder and employed fire runners; another Illinois mine was sealed in 1928 because of a fire caused by granular black blasting powder. In 1929 an

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6/ Fleming, J. R., and Koster, J. W., The Use of Permissible Explosives in the Coal Mines of Illinois: Bull. 137, Bureau of Mines, 1917, p. 8.

Oklahoma coal-mine fire was caused by granular black blasting powder shortly after this explosive had replaced a permissible explosive; and in 1932 an Illinois mine was encountering fires following blasting with granular black blasting powder. These are only a few of many available examples of relatively recent fires due to the use of granular black blasting powder.

Many fires are also caused by the use of that form of black blasting powder designated as pellet powder. In 1931 a fire occurred in an Indiana mine in spite of an inspection made by the shot firer after blasting, because he failed to see the fire on account of the dense smoke; this was duplicated in 1931 in another Indiana mine. In both instances fuse was used for firing the pellet powder, and the fires were not discovered until the second day after the shots had been fired. In other instances fires caused by pellet powder were brought under control either by fighting them directly, in some instances using respiratory protection, or by sealing the immediate area; in one case the pellet powder was fired electrically (electric squib). Recently men were suffocated when they encountered the irrespirable gases produced by a mine fire caused by blasting with pellet powder.

Mine fires are likely to be caused by any form of black blasting powder or by the improper use of permissible explosives; however, they may be prevented by the proper use of permissible explosives. "Proper" use requires (1) permissible explosives in first-class condition; (2) adequate confinement of the charge in a properly placed hole by means of incombustible stemming (if the hole is properly placed, the charge need not be excessive and will not blow out at side or back); (3) correct placement of a properly made primer; and (4) electric firing of the shot by means of electric current which is not potent (or hot) enough to ignite gas, preferably a permissible single-shot blasting unit. All of these conditions have been discussed elsewhere in this circular.

Electric Firing That Will Not Ignite Gas

Among other provisions, the Mine Safety Board of the Bureau of Mines specifies (Decision 12) that -

> 2. Each shot shall be fired separately by a permissible single-shot blasting unit ***.

The active list of permissible explosives (Report of Investigations 3259) specifies that electric firing must be done by means of a permissible-type blasting unit.

Electric firing has been specified because experience has demonstrated that the accidents resulting from electric firing, as practiced, are not as numerous as those resulting from the use of fuse or squib. This statement is confirmed by Howell's[7] study of 446 accidents in the anthracite region,

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7/ Howell, S. P., Explosives Accidents in the Anthracite Mines of Pennsylvania, 1923-27: Bull. 326, Bureau of Mines, 1931, 93 pp.

by different methods of firing some 161,000,000 shots. The number of accidents per million shots when squibs were used was 12.2; when fuse was used, 3.7; and when electric firing was employed, 1.9.

During that period the percentages of shots fired by different methods used in the anthracite region were as follows: By squib, 4.4 percent; by fuse, 53.5 percent; by electricity (electric blasting caps, electric squib, and otherwise), 42.1 percent; and by cordeau, much less than 0.01 of 1 percent.

Although electric firing unquestionably is much safer than blasting with fuse or squib, it also presents hazards peculiar to itself. Insofar as the ignition of gas is concerned a hazard arises from the ease with which, under certain conditions, an inflammable mixture of gas and air may be ignited by an arc or spark anywhere along the firing line between the source of power and the hole.

The arc or spark may be produced by a floating short circuit or from commotion at or near the face instantly following the firing of the shot, bringing the two leg wires of the electric blasting cap or wires of the firing cable in contact and breaking this contact while voltage is still on the line. The breaking of this contact may be designated as a "break spark"; the break spark will ignite gas only if the electric current is strong enough, but with permissible single-shot blasting units a break spark anywhere along the shot-firing line is not hot or potent enough to ignite gas. Permissible single-shot blasting units should therefore be used because they are safer as regards ignition of gas or dust and because they are provided with a safety contact, which guards against other types of accidents, particularly premature blasts.

Other sources of electric current, including low-capacity dry cells, which are not strong enough to ignite gas in case of a break spark are available but are not permissible because they have not been provided with a safety contact.

In order that the danger of using a source of electric current capable of producing a break spark strong enough to ignite gas may be fully understood, consider the case of the powerful 10-hole manually operated, nonpermissible magneto. If a single shot is fired the sequence of events is essentially as follows: The current is delivered to the firing circuit at the end of the stroke. In a few thousandths of a second (milliseconds) the shot-firing circuit is open-circuited either by fusing the bridge within the electric blasting cap or breaking it by firing the electric blasting cap. One or more milliseconds thereafter the shot wave set up by the shot is delivered to the surface or the face of the material blasted - for example, the face of the coal. The commotion due to the blast may or may not cause a spark from the legs of the electric blasting cap, the joints where the legs of the electric blasting cap are attached to the firing cable, or along the cable by making contact and separating. Should the bare spots on the leg wires come in contact while there is still voltage on the line -

there will be voltage on the line if the armature is rotating - a spark or arc will be produced, and if inflammable gas or dust is present it probably will be ignited. One way to reduce this chance of ignition of gas to a minimum is to place the leg wires of the electric blasting cap that extend outside the hole, the joints, and the shot-firing cable so that any commotion will not bring the two sides in contact; that is, they should be placed as far apart as is feasible. Moreover, the shot-firing cable and the leg wires of the electric blasting cap should contain the best insulation possible, and the joints should be staggered. To reduce the possibilities of spark or arc to a minimum, the joints should be staggered; there should be no bare spots on either of the leg wires of the electric blasting cap or the shot-firing cable; and one joint should be removed as far as feasible from the other joint of the line.

Another means of preventing an arc or spark in electrical blasting, utilized extensively in Europe where the equivalent of permissible explosive is employed, is to equip these powerful 10-shot blasting units or blasting units of greater capacity with a timing switch of about 0.03 second inside the case; it is thought in Europe that with reasonable precautions in connecting the cable to the leg wires of the detonator, as previously specified here, and with voltage off the line within 0.03 second the chances of commotion with possible arcing at the face or in the open air are remote.

Some Bureau of Mines engineers favor electrical firing of all shots in a mine simultaneously from outside the mine when nobody is in it.

Special Conditions Requiring Exceptions to Single-Shot Firing

Owing to the importance of certain hazards that attend the firing of shots singly under some conditions the Bureau of Mines, although in favor of firing shots singly as a rule, approves simultaneous multiple shot-firing under certain conditions:[8]

> Each shot shall be fired separately by a permissible single-shot blasting unit ***.
>
> Where more shots than one are to be fired in a working place of a mine, it is possible that under some conditions special hazards may exist in firing shots separately and inspecting between shots; in such cases simultaneous multiple shot-firing may be allowable, when permitted by the respective State mine regulations. This condition may occur in steeply pitching workings or in working places where outbursts of gas may be released by shot, or where the roof, draw slate, or coal is so friable that despite use of usual good timbering methods falls are likely to occur after a shot, making

8/ Mine Safety Board, Recommendations of the United States Bureau of Mines on Certain Questions of Safety as of February 3, 1933: Inf. Circ. 6732, July 1933, pp. 19-20.

it dangerous to inspect the roof and to connect the firing leads for the next shot. This possible exception making choice of a lesser hazard does not mean that the Bureau of Mines recommends simultaneous multiple shot-firing when it is at all practicable to fire safely one shot at a time with careful inspection for gas and roof conditions before and after each shot. When multiple shot-firing is necessitated by the conditions it should be done by firing simultaneously with standard electric detonators. The use of fuse and (or) "delayed-action detonators", so called, is dangerous because of the possibility that a delayed shot may ignite gas or a coal-dust cloud thrown out by a previous shot.

No multiple shot-firing device for use in gassy coal mines under this exception has been approved by the Bureau of Mines at the time of completing this paper (May 1933), but an investigation is being conducted to determine its feasibility; meantime, when multiple simultaneous shot-firing is necessitated by the conditions, such available multiple shot-firing device should be selected as will give as low-tension current as possible for firing not to exceed six shots at a time, with duration of current of the smallest fraction of a second. Under no circumstances should the power lines be used in blasting when men are in the mine.

Reason for Limiting the Charge per Hole to 1-1/2 Pounds

All permissible explosives have passed the exceedingly severe test of firing without ignition 1-1/2 pounds repeatedly into an inflammable mixture consisting of 4 percent natural gas, fine bituminous-coal dust, and air; this test places all permissible explosives essentially on a common competitive basis insofar as safety against ignition of gas or dust is concerned. With an adequate number of properly placed holes no hole for blasting in coal need be charged with more than 1-1/2 pounds to do the work satisfactorily; much less than 1-1/2 pounds per hole should be used when feasible, and in many mines this is the practice.

Because this limit - an arbitrary one - has been set, it is sometimes thought that the safety of a permissible explosive is no greater than that of black blasting powder or dynamite when more than 1-1/2 pounds is charged per hole. Actual tests indicate that in general at least 17 times as much permissible explosive as dynamite and at least 45 times as much permissible explosive as black blasting powder are required to place these explosives upon a safety equality; hence a large factor of safety still exists, even when considerably more than 1-1/2 pounds of permissible explosives is used per hole. If permissible explosives are confined adequately in the bore-hole and the charge is proportioned to the burden the difference in safety between 1-1/2 and 2 pounds of permissible explosive is, in all probability, not much if any greater than the difference in safety between 1 and 1-1/2 pounds of permissible explosive.

The weight of explosive per hole unquestionably should be kept as low as feasible to serve both safety and efficiency, and enough holes should be used and so placed that no hole requires more than 1-1/2 pounds; however, if this is not possible permissible explosives even if used in quantities greater than 1-1/2 pounds per hole are still much safer than dynamite or black blasting powder under similar conditions.

Amount of Stemming Required by a Charge

The Bureau of Mines recommends that the shot be confined properly with clay or other incombustible stemming; an absolute minimum length of stemming should be required. It seems essential to safety that the length of stemming in no case should be less than half the length of the charge of explosive and never less than 2 feet. In many instances the hole should be stemmed to the collar.

Time Between Firing of Nearby Shots

Gas has been ignited by rapid firing of adjacent shots, even with permissible explosives.

The hazard of rapid firing of nearby or adjacent shots lies in the release of explosive gas or dust by shots which have been fired and the firing of a later shot or shots in the presence of gas or dust, or both. Nearby shots therefore should not be fired in immediate succession unless enough time is allowed between each two shots for proper testing of the gas that may have been released by the first shot, for gas to be removed by the air currents, and for the fine, inflammable dust to settle or be rendered inert by rock-dusting or be removed by ventilation. Nearby shots are not necessarily dependent shots, though they often are; the firing of nearby dependent shots is rendered especially hazardous by possible blown-out shots and consequent ignition of gas or dust, as the latter shot may not have an adequate burden, the later shot may open a crack by the firing of the preceding one, or the charge in the later hole may even be exposed by the first shot. Nearby shots that point toward each other should not be fired together because a blown-out shot following a good shot may cause an explosion if it spends itself on the dust or gas, or both, released by the fired shot.

Nearby shots may be fired too rapidly if fired with fuse, even though the two fuses are of supposedly equal length and are lighted simultaneously; likewise, they may be fired too rapidly if delay electric blasting caps are used, even though the delay is of the same period(for example, all first delays). This is one reason why the Bureau considers the use of fuse in firing permissible explosives a nonpermissible practice and the primary reason why it considers the use of delay electric blasting caps as nonpermissible practice.

Safety in Breaking Boulders or Rock Falls with Explosives

If detached rocks or boulders are too large or too tough to be broken with a sledge they may be blasted safely by using very small charges - 1/4 to 1/2 cartridge of permissible explosive - by blockholing; a hole or holes should be drilled in the rock with bottom not too close to the side of the rock away from that in which the hole is started, and each hole should be stemmed with incombustible material to the collar. To insure greater safety, several pounds of rock dust should also be placed on the rock above the boreholes to act mainly as exterior stemming and to cool the flame of a possible blown-out shot.

In no case should bulldozing shots (mudcapped, adobe, crevice, groove, plaster, lay-on, "shock", or open shots) be permitted in any coal mine, as they are extremely dangerous, even where no explosive gas is present. Shots of this type should be prohibited, partly because they are inefficient but primarily because they ignite gas or dust so readily that even the use of rock dust or other incombustible material over the charge does not insure that degree of safety to which men and mine are entitled. Certainly no shots of this type should ever be fired with the working shift in the mine.

Harrington and Owings 9/ have called attention to a number of explosions and other accidents from mudcapped shots in coal mines. Although high explosives other than permissible explosives have usually been involved bulldozing shots of permissible explosives have caused explosions.

Bulldozing of rock or coal in place is just as hazardous as bulldozing detached rocks or boulders; to insure safety in coal mines blasting should be done only with permissible explosives properly confined in boreholes.

EXPERIENCE IN USE OF PERMISSIBLE EXPLOSIVES

The following sections of this report include an estimate of the experience in the use of permissible explosives, statements regarding the quantity of explosives used for blasting in coal mines, the percentage (estimated) of coal blasted by permissible explosives, the percentage (estimated) of shots of permissible explosives fired in hazardous places, such as gassy or dusty mines or gassy or dusty sections or places in mines, by comparison with other explosives, and the number of accidents caused directly by the explosives.

The record of gas or dust ignitions from black blasting powder, dynamite, and permissible explosives is taken from such reports as are available to the Bureau of Mines. These data show that no permissible explosives when used in a permissible manner have caused an ignition of either gas or dust and that no gas ignition has occurred from permissible explosives unless two or more nonpermissible conditions existed during their use.

9/ Harrington, D., and Owings, C. W., Explosions and Other Accidents from Mudcapped Shots in Coal Mines: Inf. Circ. 6158, Bureau of Mines, 1929, 5 pp.

Explosives Used and Explosives Accidents, 1912-32

In 1902 the first "short-flame" explosives were used in the United States, the quantity aggregating 11,300 pounds. In 1909 the first permissible explosives were used, the quantity sold reaching 9 million pounds; in 1913 more than 20 million pounds were used in coal mines. Comparison of the quantities of black blasting powder, high explosives (other than permissible explosives), and permissible explosives sold for use in coal mines from 1912 to 1932 show that sales of black blasting powder have always greatly exceeded sales of either high explosives or permissible explosives. Sales of black blasting powder declined slowly from 1912 to 1915, reached a maximum of 235,750,300 pounds in 1917, and fluctuated with the erratic production of coal from 1918 to 1923, standing at 175,488,750 pounds in the latter year; since 1923, however, they have decreased rapidly and rather uniformly, irrespective of whether the production of coal remained high, as it did from 1923 to 1929, or whether the production declined markedly, as it has done since 1929. In 1929 sales of black blasting powder aggregated 102,060,255 pounds and in 1932; 56,311,975 pounds.

From 1912 to 1932 sales of high explosives (other than permissible explosives) for use in coal mines ranged from 37,828,979 pounds in 1923 to 16,173,014 pounds in 1932. Since 1926 the quantity decreased steadily from 35,228,899 pounds. A large proportion of the high explosives (mostly dynamite) now consumed in coal mining in the United States is used for rockwork in the Pennsylvania anthracite mines.

Sales of permissible explosives for use in coal mines of the United States increased from 1912 until 1926, when a maximum of 65,143,319 pounds was reached, and declined steadily to 31,532,816 pounds in 1932. Sales have fluctuated with total coal production, showing a decidedly upward trend until 1929. Since 1915 the quantity of permissible explosives used in coal mines has exceeded the quantity of high explosives and since 1924 has been about twice as much.

Coal production increased markedly from 1914 to the 1918 peak of 678,211,904 short tons, was erratic until 1926, and declined markedly from 1926 to 1932, except for 1929. In 1932 the production was 359,565,000 short tons. Obviously sales of explosives for use in coal mines have kept close pace with coal production.

Table 1 shows the number of coal-mine fatalities in the United States involving explosives for the period 1916-31. These fatalities show a general downward trend from a maximum of 241 in 1919 (very high due to the "powder" explosion in the Baltimore No. 2 tunnel in Pennsylvania, which caused 92 fatalities) to a minimum of 47 in 1931; in general, the downward trend of fatalities from explosions coincides fairly closely with the reduction in the use of black blasting powder, although other factors have some significance.

Table 1. - Number of coal-mine fatalities in the United States involving explosives, 1916-31

Year	Fatalities due directly to explosives			Fatalities from gas and dust explosions due to explosives or blasting	Total
	Underground	On surface	Total		
1916	146	2	148	32	180
1917	110	13	123	26	149
1918	135	11	146	40	186
1919	206	1	207	34	241
1920	131	11	142	48	190
1921	142	5	147	30	177
1922	93	9	102	67	169
1923	115	10	125	39	164
1924	99	3	102	25	127
1925	102	6	108	93	201
1926	96	2	98	54	152
1927	110	5	115	21	136
1928	74	2	76	16	92
1929	88	5	93	7	100
1930	78	4	82	10	92
1931	40	1	41	6	47
Total	1,765	90	1,855	548	2,403
Average	110	6	116	34	150
Percentage	73.5	3.7	77.2	22.8	100.0

The reduction in number of fatalities from coal-mine blasting is due largely to a reduction in underground fatalities caused directly by explosives but is influenced favorably by a large proportional, but small aggregate, reduction in fatalities on the surface directly attributable to explosives and by a substantial reduction in fatalities attributable to dust and gas explosions initiated by explosives or blasting. (See table 1.) Underground fatalities in recent years have been decreased materially owing to the use of proportionately less black blasting powder and dynamite, greater care in transportation and handling of all explosives, increased use of electric blasting, knowledge of the hazards of electric blasting (it does present hazards, particularly in connection with premature shots) and how to guard against them, safer blasting methods, and probably better instruction and supervision.

The reduction in the number of fatalities from gas and dust explosions caused by explosives and blasting in recent years is due partly to increased use of permissible explosives, wider use of permissible explosives in dangerous mines or in hazardous places (gassy or dusty, or both) in mines, adoption

of rock-dusting in some of the more hazardous mines or sections of mines, more intensive and more intelligent inspection given the operating practices of many dangerous mines, better blasting practices, and efficient ventilation and gas testing resulting from improved instruction and supervision.

Machine mining increases the efficiency of explosives considerably, permits the use of relatively small charges, and therefore contributes largely to a reduction in the quantity of explosives used per 1,000 tons of coal mined; this in turn aids safety in the use of explosives, particularly where permissible explosives fired electrically are employed skillfully and substituted for black blasting powder and dynamite.

Mechanical loading, however, often tends to increase the hazards of blasting by interfering with efficient ventilation, releasing firedamp rapidly through rapid extraction, creating a hazardous dusty condition, firing an increased number of shots daily in the same place, using dependent shots or delay electric blasting "caps" or both, and usually requiring blasting during the main working shift. The most intensive and competent supervision of the blasting operation is required to prevent gas or dust ignitions and premature blasts when firing electrically, which is hazardous with mechanical loading because the latter generally utilizes electricity as power close to places where holes are charged and blasted.

The proportionately greater increase in the use of permissible explosives in the anthracite region of Pennsylvania compared with bituminous-coal mines in the United States is shown in table 2 for 1917-32. Anthracite mines consumed 17 percent of the total amount of permissible explosives used in coal mining in 1917, a minimum of 15.9 percent in 1920 and a maximum of 41.2 percent in 1932; use of permissible explosives in bituminous mines fluctuated from a maximum of 76.0 percent of the total in 1922 to a minimum of 56.7 percent in 1932. It is significant that of the total quantity of permissible explosives manufactured 76.6 percent was used in coal mines in 1917 and 97.9 percent in 1932; the explanation of this is that special explosives and explosives similar to permissible explosives, but not so denominated, are now used to a considerable extent in place of permissible explosives in metal and nonmetal mining and in railroad, quarrying, and other construction work.

Table 2. - Percentage of total quantity of permissible explosives manufactured which was used in anthracite and bituminous coal mines, 1917-32 1/

Year	Percentage used in			Year	Percentage used in -		
	Anthracite mines	Bituminous-coal mines	All coal mines		Anthracite mines	Bituminous-coal mines	All coal mines
1917	17.0	59.6	76.6	1925	20.7	68.9	89.6
1918	16.6	64.3	80.9	1926	28.2	68.0	96.2
1919	22.1	62.4	84.5	1927	29.4	66.0	95.4
1920	15.9	67.9	83.8	1928	31.8	62.5	94.3
1921	29.0	63.5	92.5	1929	30.8	65.3	96.1
1922	16.6	76.0	92.6	1930	31.9	65.4	97.3
1923	24.2	69.9	94.1	1931	37.6	60.1	97.7
1924	30.7	63.9	94.6	1932	41.2	56.7	97.9

1/ This table does not indicate the relative amount of permissible and nonpermissible explosives used in coal mines, or the relative amounts of coal produced by the different kinds of explosives.

EXPLOSIVES USED IN ANTHRACITE REGION OF PENNSYLVANIA, 1917-32

The record of the amount of black blasting powder, high explosives, and permissible explosives sold for use in the anthracite mines of Pennsylvania per 1,000 short tons of anthracite mined shows some highly significant trends. The quantity of black blasting powder sold per thousand tons of anthracite has decreased from a maximum of 359 in 1917 to a minimum of 152 pounds in 1932; the quantity of high explosives has increased from 162 in 1917 to a maximum of 323 pounds in 1927 and 1929 and has decreased since 1929 to 226 pounds in 1932; the quantity of permissible explosives has increased steadily from 73 in 1917 to a maximum of 269 pounds in 1932, although a reduction is noted for 1930. The trend in the use of black blasting powder shifted upward in 1929 and 1930, owing largely to the increased use of pellet powder, but since 1929 has been markedly downward. The downward trend in use of high explosives per thousand short tons of anthracite since 1929 is due partly to the substitution of more powerful types of permissible explosives in chute blasting but chiefly to a progressive reduction of rock work.

The increased use of permissible explosives in the anthracite region is an encouraging feature of the story of permissible explosives and of progress in safety in anthracite mining. By 1926 the quantity of permissible explosives per 1,000 pounds in anthracite mining exceeded that of black blasting powder; by 1931 it equaled the quantity of high explosives used, and in 1932 it exceeded the quantity of high explosives used. Its use has increased because of more general recognition of the adaptability of permissible explosives for all blasting purposes, a much greater safety in gassy places, fewer gas ignitions compared with black blasting powder, a substantial increase in electric blasting because of its greater safety, and the greater strength inherent in certain permissible explosives of the gelatine type. Many anthracite collieries use permissible explosives exclusively in blasting coal, and many collieries - in some instances all the collieries of a State inspection district - fire all shots electrically. Incidentally, electric firing is the only logical method of firing shots where closed lights are used exclusively and an effort is made to keep matches and smoking out of the mine, inasmuch as the use of matches in smoking is a major cause of gas explosions in the anthracite region. However, use of electric firing and electric cap lamps and failure in the necessary measures to exclude matches and smoking are certain to result in gas ignitions in any coal mines in which gas occurs.

From 1917 to 1932 anthracite production varied materially, but the trend was generally downward - 99,611,811 short tons in 1917 to 49,855,221 in 1932, with very low yearly production in 1922 and 1925. The low production in these years was made with much less explosives per 1,000 tons, possibly because development work was discontinued temporarily, considerable coal, previously blasted, was drawn from full breast, and other means were adopted to decrease cost.

The percentage of anthracite mined with permissible explosives was estimated on the following basis: That 25 percent of the high explosives was used in coal mining including chute blasting, that virtually all black blasting powder and "permissibles" were used in blasting coal, and that in blasting coal 1 pound of permissible was equivalent to 1 pound of high explosives and 1-3/4 pounds of black blasting powder. This estimate shows a progressive and rather uniform upward trend in the anthracite obtained through use of permissible explosives from 23 percent in 1917 to 65 percent in 1932. With full recognition of all the hazards attending the storage, handling, and use of black blasting powder, the greater hazard of chute blasting with dynamite than with permissible explosives, and the assurance that electric blasting can be accomplished safely, the percentage of anthracite mined with permissible explosives fired electrically probably will increase steadily above the estimated 65 percent in 1932. The yearly variations are shown in table 3.

TABLE 3. - Estimated quantity and percentage of explosives used by classes in anthracite and bituminous-coal mines in the United States, 1917-32

Year	Pounds per thousand tons			Percent of explosive sold for use			Percent of coal blasted with permissible explosive	Production, short tons
	Permissible	High explosive	Black blasting powder	Permissible	High explosive	Black blasting powder		
	Anthracite mines							
1917	73	162	359	12.3	27.3	60.4	23	99,611,811
1918	77	169	307	13.9	30.6	55.5	26	98,826,084
1919	98	202	289	16.6	34.3	49.1	31	88,092,201
1920	95	215	258	16.7	37.9	45.4	32	89,598,249
1921	132	252	264	20.4	38.9	40.7	38	90,473,451
1922	132	228	231	22.3	38.6	39.1	41	54,683,022
1923	156	274	233	23.5	41.3	35.2	43	93,339,009
1924	192	305	229	26.5	42.0	31.5	48	87,926,862
1925	195	288	206	28.3	41.8	29.9	51	61,817,149
1926	226	310	201	30.7	42.0	27.3	54	84,437,452
1927	235	323	187	31.5	43.4	25.1	55	80,095,564
1928	251	302	182	34.1	41.1	24.8	58	76,734,000
1929	262	323	201	33.3	41.1	25.6	57	73,828,195
1930	247	311	197	32.7	41.2	26.1	57	69,384,837
1931	262	261	173	37.7	37.5	24.8	61	59,645,652
1932	269	236	152	41.0	35.9	23.1	65	49,855,221
	Bituminous-coal mines							
1917	47	22	362	10.9	5.1	84.0	19	551,790,563
1918	51	25	321	12.8	6.3	80.9	21	579,385,820
1919	52	25	275	14.8	7.1	78.1	25	465,860,058
1920	64	32	346	14.5	7.2	78.3	24	568,666,683
1921	63	27	280	17.0	7.3	75.7	28	415,921,950
1922	78	31	337	17.5	6.9	75.6	29	422,268,099
1923	75	22	272	20.3	6.0	73.7	33	564,564,662
1924	73	20	249	21.3	5.9	72.8	34	483,686,538
1925	77	18	230	23.7	5.5	70.8	37	520,052,741
1926	80	16	208	26.3	5.3	68.4	40	573,366,985
1927	81	16	189	28.3	5.6	66.1	43	517,763,352
1928	77	14	177	28.7	5.2	66.1	43	492,755,000
1929	76	15	163	29.9	5.9	64.2	45	534,988,593
1930	75	14	154	30.9	5.8	63.3	46	467,526,299
1931	65	12	152	28.6	5.3	66.1	43	382,089,396
1932	60	15	159	25.6	6.4	68.0	40	309,709,872

EXPLOSIVES USED IN BITUMINOUS-COAL MINES IN THE UNITED STATES, 1917-32

The trend of the quantity of the three classes of explosives used per 1,000 short tons of bituminous coal mined from 1917 to 1932 (table 3) was: For black blasting powder, a maximum of 362 pounds in 1917, an erratic decrease to 272 pounds in 1923, a decided decrease to 154 pounds in 1930, and a slight increase from 152 pounds in 1931 to 159 pounds in 1932; for high explosives, an upward tendency from 22 pounds in 1917 to 32 pounds in 1920, a decrease to 12 pounds in 1931, and a slight increase to 15 pounds in 1932; for permissible explosives, an erratic increase from 47 pounds in 1917 to a maximum of 81 pounds in 1927 and a decrease, rapid after 1930, to 60 pounds in 1932.

The quantity of permissible explosives used in the coal mines of the United States has never equaled half the quantity of black blasting powder so used, although it approached the quantity of black blasting powder in 1930. Permissible explosives in general gained steadily on high explosives from 1917 to 1932, obviously because of the substitution of the former in rockwork and elimination of much of the rockwork in which high explosives would ordinarily be employed. Black blasting powder retains its supremacy in quantity used, largely because it costs less and can be (and generally is) fired with fuse or squib or even fuse and caps, each usually costing less than electric blasting caps, and because most operators believe it will produce more lump coal or lumps of greater solidity and less slack coal than permissible explosives. Permissible explosives, however, if used skillfully, have no peer in producing lump coal in machine-mined places; elsewhere they are either on a par with black blasting powder or but little less efficient. Even under unfavorable conditions as to quality of coal produced the far greater safety of permissible explosives is a small price to pay for any relatively slight uneconomic feature that may be involved. Permissible explosives are used largely for blasting machine-mined coal, for blasting in gassy or dusty places, and for blasting top, bottom, or intermediate bands of rock in coal mining. The great range of the strength characteristics of permissible explosives favors their use for the variety of blasting conditions encountered in bituminous and lignitic mines in the United States.

The percentage of bituminous coal mined with permissible explosives was estimated on the following basis: That all coal was mined with either permissible explosives or black blasting powder, that 1 pound of permissible explosive was equivalent to 1-3/4 pounds of black blasting powder, and that the permissible explosives used in blasting rock were more than compensated by the high efficiency obtained with them in blasting machine-mined coal. This estimate shows that in 1917 about 19 percent of the bituminous coal was blasted with permissibles and that the percentage rose rather steadily to about 46 percent in 1930 and then declined to approximately 40 percent in 1932.

Apparently permissible explosives have lost ground since 1930 in the bituminous mines in the United States but have gained in the anthracite region in Pennsylvania.

PERMISSIBLE EXPLOSIVES USED UNDER HAZARDOUS CONDITIONS

Permissible explosives are more efficient, pound for pound, than black blasting powder, and where used under similar conditions the proportion of coal blasted with permissibles is correspondingly greater for both anthracite and bituminous-coal mines.

It is important, however, to stress the fact that a large percentage of the coal from gassy or dusty mines is blasted with permissible explosives. The opportunities for gas or dust ignitions are therefore more frequent with permissible explosives than with either black blasting powder or dynamite; it is estimated that permissible explosives as now used have at least 90 percent of these maximum opportunities to ignite gas or dust. In view of the hazardous conditions which generally attend the use of permissible explosives and the nonpermissible conditions under which they frequently are used it would not be surprising if a large proportion of the ignitions of gas or dust in coal mines were caused by permissible explosives; such, however, is not the case, as stated in the introduction to this circular.

The Bureau of Mines has a record of 117 coal-mine explosions caused by explosives in the United States from 1908 through 1932. Of these 12 were in anthracite mines and 105 in bituminous-coal mines; only 13 are chargeable to permissible explosives used in a nonpermissible manner and 2 to permissible blasting devices used likewise. In these 117 explosions 1,136 men were fatally injured and 235 men were nonfatally injured.

Of the 13 explosions known to have occurred when permissible explosives were used (all in a nonpermissible way) 7 were gas explosions, 1 was a dust explosion, 4 were gas and dust explosions, and 1 was an air blast caused by a greatly overcharged shot.

In all instances in which gas or dust was ignited the explosives were fired in gas, but in each one of these instances as well as those in which dust was ignited additional nonpermissible conditions of use - sometimes as many as three - were in effect. In 4 instances the shots were obviously likely to blow out, in 1 the hole was not stemmed to the collar, in 1 combustible stemming was used, in 6 the shot was overcharged, in 1 and possibly 2 adjacent shots were fired rapidly, using electric firing. In 1 instance adjacent shots were fired rapidly by fuse, in 1 tests were not made for gas, in 5 - all gas ignitions - a nonpermissible shot-firing device was used and possibly the gas was ignited by the current used for firing electrically, in 3 live currents were used for firing, and in 2 the charge was not in a borehole (that is, they were "bulldozing" shots).

SUMMARY

1. According to results of gallery tests the factor of safety against ignition of inflammable gas by permissible explosives over black blasting powder is at least 45 and over common dynamite at least 17.

2. Experience has shown that no permissible explosives when used in a permissible manner have caused ignition of either gas or dust in any coal mine in the United States and that no gas ignition has occurred from permissible explosives except where 2 to 4 nonpermissible conditions existed during their use.

3. Coal-mine fatalities from explosives show a decidedly general downward trend, from a maximum of 241 in 1919 to a minimum of 47 in 1931. The downward trend coincides fairly closely with the reduction in sales of black blasting powder. Gas and dust explosions due to explosives caused 22.8 percent of the fatalities from explosives in the period 1919-31, inclusive.

4. The reduction in the number of fatalities from gas and dust explosions caused by explosives or blasting is due chiefly to the increased use of permissible explosives, particularly in hazardous places (gassy or dusty, or both); adequate rock-dusting in some of the more hazardous mines or sections; greater care in transportation and handling of all explosives; increasing use of electric blasting, with a knowledge of its hazards and how to guard against them; and better blasting practice (including ventilation and gas testing) resulting from improved instruction and supervision.

5. To insure maximum safety in the use of permissible explosives, the following should be observed:

(a) Permissible explosives should be fired electrically, as accidents are not as numerous when they are so fired as when fuse and blasting caps are used.

(b) The electrical current used for firing should not be potent enough to ignite inflammable gas in the event of a break spark.

(c) The safest blasting system is that of taking the explosive into the mine and loading it into the blasting holes after the working shift has left the mine and of firing all shots simultaneously from outside when nobody is in the mine.

(d) Permissible explosives should not be fired in a dangerous percentage of firedamp.

(e) Ignition of inflammable coal dust should be guarded against by the use of water on the cutter chain and on and at the face and by rock-dusting.

(f) Charges should be properly confined in boreholes and stemmed with incombustible material.

(g) The explosive should be in first-class condition when used; in other words, deteriorated explosives should not be used.

(h) The charge of permissible explosive per hole should be limited to 1-1/2 pounds as a maximum.

(i) The face should be properly prepared for blasting.

(j) The primer should be properly prepared and located in the hole.

(k) There should be an adequate number of properly placed holes, each of adequate gage, to permit ready placement of the charge.

(l) Competent shot firers only should charge and tamp holes; the competence of such shot firers should be attested by a State certificate as mine examiner, fireboss, or foreman. As an alternative the shots may be charged and the holes tamped under the immediate supervision of a man so qualified. The shot firers should at all times be under close supervision and discipline.

(m) Both before and after charging a hole the shot firer should assure himself by a suitable test, such as one employing a permissible flame safety lamp, that no explosive gas is present; if he finds gas in quantities that indicate danger the hole should not be charged, and if he finds gas after the hole is charged the shot should not be fired. Holes may be charged and shots fired, however, after the gas has been removed by ventilation.

(n) The methods of firing shots in coal mines in the order of preference are: Charging by shot firers after the working shift and firing from the surface with all men out of the mine; charging and firing by shot firers when all other men are out of the mine; charging and firing by shot firers on the off shift; charging and firing by shot firers on the regular or main shift; charging by miners during the shift and firing by shot firers when all other men are out of the mine; charging and firing by miners at the end of the shift or at some specified time during the shift, such as midday. The method of charging and firing by miners at any time during the shift, although practiced in many mines, is neither safe nor efficient and therefore is the least preferred.

(o) Except when firing from outside the mine, shots preferably should be fired singly by a permissible single-shot blasting unit, and nearby shots should not be fired in rapid succession.

(p) Permissible shot-firing units (unless firing is done from the surface) should be used by either shot firers or miners. Such units provide electric current that is not potent enough to ignite gas or break sparks, and they do not have exposed terminals which may cause premature blasts.

(q) The use of the power lines for blasting in coal mines is distinctly hazardous unless utmost care is taken; the power wires should not be used for blasting in coal mines when men are in the mine.

(r) Shot firers or miners should not return to the face too promptly after firing, as a single shot may under abnormal conditions explode twice, the smoke and dust may so obscure the roof and face that approach is hazardous, or the noxious gases and fumes may not have been adequately diluted by ventilation and rendered harmless.

(s) The electrical short circuit on the legs of the electric blasting cap should not be removed until just before the leg wires are connected to the shot-firing cable.

(t) The shot-firing cable should be at least 100 feet long; it should be kept in good repair and should be detached from the source of the firing current at all times except at the instant that a shot is being fired.

(u) Shots should be properly guarded, and the intention to fire should be declared in a standardized manner.

(v) Bulldozing or other open, unconfined shots should not be fired in any coal mine.

6. Misfires may be reduced by selecting proper explosives and electric blasting caps, by keeping them in first-class condition through proper storage until used, by using the explosive before it has aged unduly, and by correct priming, charging, and firing of the explosive.

7. In this circular an attempt has been made to estimate the present status of the use of permissible explosives, compared with their use in previous years and with the use of nonpermissible explosives for anthracite, bituminous, and all coal mines.

8. Since 1915 the quantity of permissible explosives used in coal mines has always exceeded that of high explosives, and since 1924 about twice as much permissible explosive as dynamite has been used. Sales of black blasting powder for mining purposes have always greatly exceeded sales of either high explosives or "permissibles."

9. Permissible explosives are not unduly sensitive to frictional impact but are sensitive enough to detonation not to promote misfires; their sensitiveness may be impaired by improper storage.

10. Permissible explosives of classes 1 and 6 are not ignited readily, do not burn vigorously after ignition, and will not explode promptly upon ignition. The black blasting powders, however, do not exhibit any of these favorable characteristics.

11. The proper brand of permissible explosive should be selected for use. Permissible explosives have a wide range of physical characteristics and are adaptable to essentially every type of blasting in coal mines. The proportion of prepared sizes of coal produced depends upon the skillful application of a suitable brand of permissible explosive.

12. Class A or B (basis, volume of poisonous gases), preferably class A, permissible explosives should be used in poorly ventilated places.

13. The hazard of gas ignition can be reduced substantially by the use of "sheathed" permissible explosives.

14. Neither dynamite nor any type of black blasting powder should be used for any purpose in any underground coal mine; suitable types of permissible explosives are available to perform any explosive function in any coal mine. The vastly superior safety qualities of permissible explosives over those of any dynamite or black blasting powder now available make it imperative that only permissible explosives be used in underground coal mines.

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METHODS OF DEVELOPMENT AND PILLAR EXTRACTION IN MINING THE PITTSBURGH COAL BED IN PENNSYLVANIA, WEST VIRGINIA, AND OHIO[1]

By J. W. Paul[2] and L. N. Plein[3]

INTRODUCTION

The United States Bureau of Mines has made a study of roof control in 68 mines operating in the Pittsburgh coal bed in Pennsylvania, West Virginia, and Ohio. In making these studies, much information was also obtained on various exact methods of development and pillar extraction as practiced by this large group of mines. It is proposed in this paper to describe these methods, by which this group of mines in 1931, a year of reduced industrial activity, produced 28,123,000 tons of bituminous coal - 7.4 percent of the national production for the same year and an amount equal to the combined production of the coal mined in the States of Alabama, Colorado, New Mexico, Utah, and Wyoming. The results of these studies, insofar as they pertain to the prevention of injuries from falls of roof, have been published by the Bureau of Mines. In this paper only a few mines typical of each system will be described; however, should the reader be interested in other details with respect to these mines, he may refer to the following publications:

T.P. 522, 1932, 43 pp. Falls of roof and coal in mines operating in the Pittsburgh coal bed in Marion and Monongalia Counties, W. Va., by J. W. Paul and J. N. Geyer.- A study of nine mines in these two counties of northern West Virginia.

T.P. 534, 1932, 34 pp. Falls of roof and coal in mines operating in the Pittsburgh coal bed, Panhandle district, W. Va., by J. W. Paul and J. N. Geyer.- A study of six mines in Brooke, Ohio and Marshall Counties.

T.P. 541, 1932, 97 pp. A study of mine roof of the Pittsburgh coal bed in the Pittsburgh mining district, by J. W. Paul and L. N. Plein.- A study of fifteen mines in Allegheny and Washington Counties, Pennsylvania.

T.P. 547, 1933, 23 pp. Falls of roof in mines operating in the Pittsburgh coal bed, West Virginia, by J. W. Paul and J. N. Geyer.- A résumé of twenty mines studied in the Fairmont and Panhandle Districts.

T.P. 550, 1933, 31 pp. A study of roof in Pennsylvania mines contiguous to the Monongahela River, by J. W. Paul and J. G. Calverley.- A study of seven mines principally in Greene and Washington Counties, Pennsylvania.

T.P. 563, 1935, 34 pp. A study of mine roof in the Coking District of Western Pennsylvania, by J. W. Paul and L. N. Plein.- A study of twelve mines principally in Fayette County.

R.I. 3070, 1931, 32 pp. A study of falls of roof and coal in mines in the Number 8 Field of Eastern Ohio, by J. W. Paul and L. N. Plein.- A study of seven mines in Belmont, Harrison and Jefferson Counties.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6872."

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R.I. 3110, 1931, 30 pp. A study of falls of roof and coal in mines of Harrison County, W. Va., by J. W. Paul and J. N. Geyer.- A study of five mines in this county.

R.I. 3113, 1931, 77 pp. Roof support in coal mines in the Irwin, Greensburg and Latrobe Basins, Westmoreland County, Pa., by J. W. Paul, H. Tomlinson, and S. J. Craighead.- A study of twelve mines.

I.C. 6200, 1929, 21 pp. Method and cost of mining the Pittsburgh or No. 8 Coal Bed in a 100 percent mechanized mine in Eastern Ohio, by W. F. Hazen and E. J. Christy.- A study of methods and costs of a mine using mechanical loading.

I.C. 6208, 1929, 21 pp. Method and cost of mining the Pittsburgh or No. 8 Coal Bed in four Eastern Ohio mines, by J. W. Paul and H. Tomlinson.- A study of four mines in Belmont and Harrison Counties.

The Pittsburgh coal was one of the first beds of bituminous coal to be extensively mined in this country, and therefore the literature on its geology, structure, composition, and method of extraction is rather lengthy; it is described in publications of Government bureaus, transactions of technical societies, and in the technical press. This paper is not intended to be a résumé of all that has been published, nor a history of the changes in methods of mining.

Most of the mines used as the basis of discussion in this report were carefully studied during the calendar years 1928 to 1932, inclusive. The manuscript was completed early in 1933 but lack of printing funds has delayed its publication. Some of the mines described have since modified their methods but it is believed that the paper still gives a good picture of mining practice in the Pittsburgh coal bed.

THE PITTSBURGH COAL BED

Because of its quality, persistence, thickness, areal extent, and uses as a steam, coking, and gas-making coal, the Pittsburgh bed may well be classed as one of the world's greatest coal deposits. Because of its proximity to industrial markets and because it is served by excellent systems of rail and river transportation, it probably is being exhausted more rapidly than any other bed of coal.

The bed occurs and is mined in four States: Pennsylvania, West Virginia, Ohio, and Maryland. Its general extent is shown in figure 1, which, however, does not include the Georges Creek field of Maryland. In 1931 the production from the Pittsburgh bed of coal was 80,000,000 tons of coal or 20.8 percent of the national production of bituminous coal for that year. In the same year the combined production of Illinois and Kentucky, our third and fourth largest coal-producing States, was 85,000,000 tons. The areal extent of the Pittsburgh coal bed has been estimated at 5,729 square miles with minable reserves of approximately 22 billion tons.[4]

Much has been published on the geology of the Pittsburgh coal bed, the rocks with which it is associated, the topography of the region, its chemical composition, and other features, and for detailed information with respect to these matters the reader is referred to publications of the United States Geological Survey, United States Bureau of Mines, the geological surveys of Pennsylvania, West Virginia, Ohio, and Maryland, Transactions of the American Institute of Mining and Metallurgical Engineers, other technical and scientific institutes, and the technical press.

4 White, I. C., Ashley, G. H., and Bownocker, J. A., The Pittsburgh Coal Bed: Trans. Am. Inst. Min. and Met. Eng., vol. 74, 1926, pp. 481-503; discussion, 503-506.

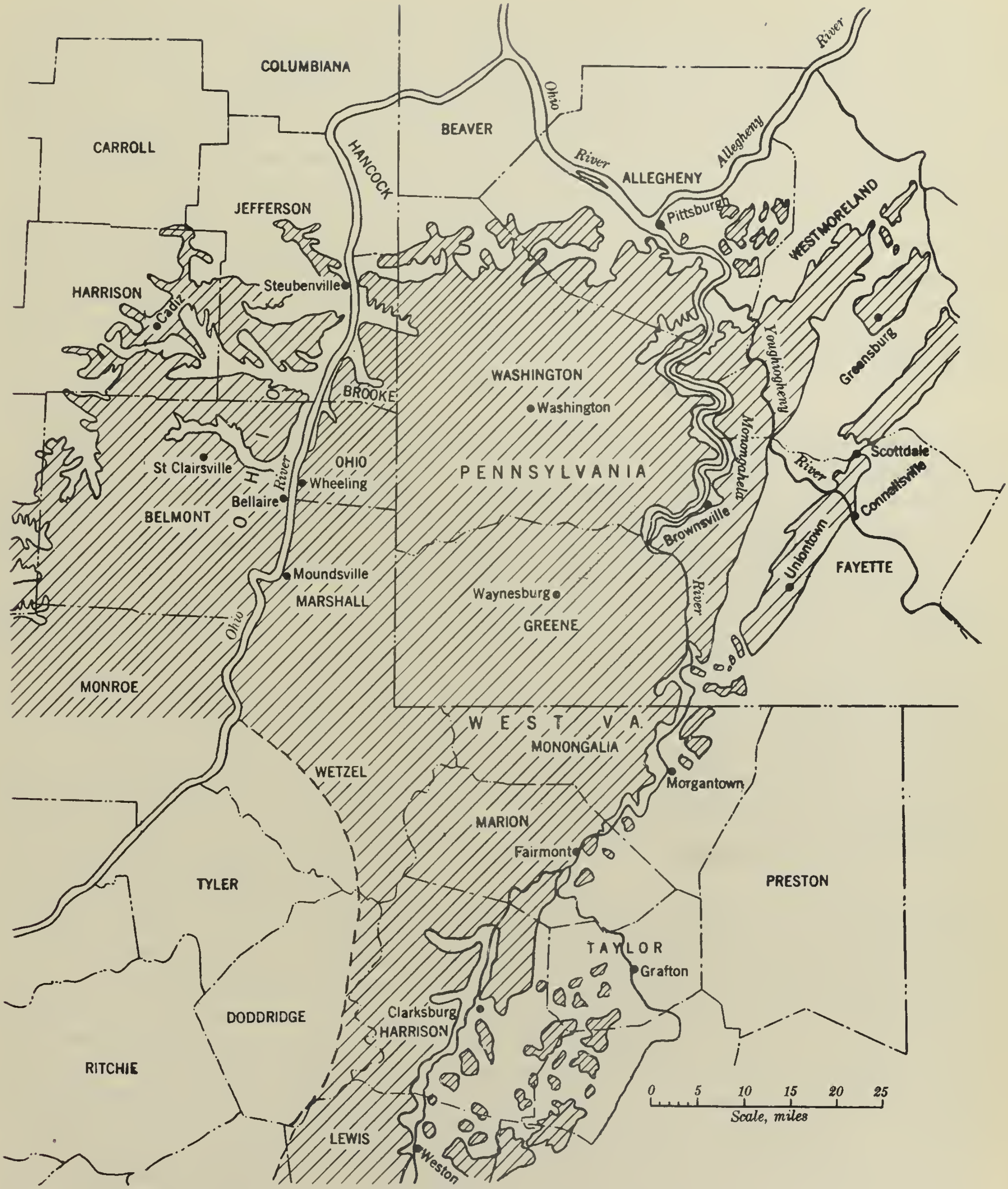

Figure 1.- Map of Pittsburgh coal field in Pennsylvania, West Virginia, and Ohio.

Briefly, the Pittsburgh coal bed lies in a very broad structural basin or trough whose major axis has a northeast and southwest trend through the city of Pittsburgh. West of this line the coal bed has a general dip to the southeast, and east of this line the bed has a general dip to the northwest but is thrown into a succession of minor folds which in general are parallel to the major axis of the basin. Structure determines the coal-bed dip, which of course, has an important bearing on such mining problems as haulage and drainage. The dip at the mines studied varied from 0 to 6 percent in a few mines, the average being 1.3 percent: Therefore, as compared to coal beds in the anthracite region of Pennsylvania or in the Rocky Mountain States, the Pittsburgh coal bed may be considered practically flat. In the discussion of the various types of mining, unless otherwise stated, it should be understood that the coal bed is practically flat and that the dip has no effect upon methods of development, haulage, or drainage, although in some cases minor changes in details may result according to the prevailing dip of the coal bed. In a few cases where the dip reaches a maximum of 6 percent, the method of development is altered. These changes will be discussed at the appropriate point in this paper.

Topographically the area varies from gently rolling country, as in eastern Ohio and many parts of Pennsylvania and West Virginia, to rather hilly country with V-shaped valleys, particularly along the Ohio, Monongahela, and Youghiogheny Rivers and their major tributaries. In the present mining districts the presence or absence of the coal bed is determined by the amount of erosion which has taken place in recent geologic time.

The Pittsburgh coal bed has certain physical characteristics which might be termed its key "fossils." These are the two characteristic binders (sometimes only one and sometimes three), which usually occur in the lower half of the bed, and its immediate roof, which normally is composed of a draw slate and a roof coal. In thickness the bed varies from 51 to 114 inches. In the Panhandle districts of Pennsylvania and West Virginia and in eastern Ohio the coal is about 60 inches thick. Going eastward into Allegheny and Washington Counties it is about 66 inches thick. In Westmoreland County it is 84 inches thick, in Fayette County 96 inches thick, and in northern West Virginia 108 to 120 inches thick. The well-defined face and butt cleats of the bed are an important factor in the method of development, which in most of the mines studied is planned so that rooms are advancing on the face cleats. The amount of cover varies from outcrop to a maximum of 900 feet with an average cover of 300 feet for the mines studied.

No attempt is made in this paper to give a detailed discussion of the stratigraphy of the rocks overlying the Pittsburgh bed of coal, as that subject is fully covered in publications of the United States Geological Survey and reports of the geological surveys of the various States. The amount of cover at the various mines indicates that in ascending order the strata of the Monongahela and Dunkard formations compose the overlying beds of rocks. The Monongahela formation extends from the base of the Pittsburgh coal bed to the top of the Waynesburg coal bed. The formation varies from 300 to 400 feet in Pennsylvania and northern West Virginia to 200 feet on its western outcrop in Ohio. In ascending order it contains the following coal beds: Pittsburgh, Redstone, Sewickley, Uniontown, and Waynesburg. The Pittsburgh bed is minable wherever it exists, but the other beds are variable and are only mined locally. The most important lithologic member of this formation is the Benwood limestone, which occurs in the middle of the formation and varies from 70 to 120 feet in thickness. Shales, sandy shales, and sandstones compose the remainder of the formation. The only other important detail of stratigraphy is the gradual change in the character of the rocks forming the immediate roof over the Pittsburgh coal bed. As the formation is followed westward, there is a change from sandstones and sandy shales, to shales and calcareous shales, to calcareous shales and limestone.

Where the cover exceeds 300 to 400 feet the Dunkard formation forms part of the overlying strata in addition to the Monongahela formation. The Dunkard formation is variable in thickness and is composed of sandstones, shales, and limestones. It is divided into an upper and lower member termed the "Greene" and "Washington" groups, respectively.

In discussing each group of mines, further details will be given as to cover, thickness of coal bed, and character of immediate roof.

METHODS OF DEVELOPMENT

The method of development of any coal bed is determined, to some extent, by dip, cover, direction of cleats, if present, location of transportation facilities with respect to area to be mined, and character of immediate roof; however, the method of pillar extraction to be used is one of the most important factors in making decisions upon the finer points of development. The following is a classification of the methods used by the 68 mines studied. This classification is based primarily on the method of pillar extraction, and, secondarily, for simplification in presentation, on a geographic basis. All of the mines studied can be classified in one of the following groups:

Class A.- All coal is mined on advance; this means that room pillars are not extracted. This is common practice in the No. 8 field of Ohio and the Panhandle district of West Virginia. Type A.

Class B.- Part of coal is mined on advance and pillars mined on retreat. This is common practice in Pennsylvania and the Fairmont district of West Virginia. This classification can be subdivided further.

I. Full advance with room pillars mined as butt entry develops. Used by mines in the Panhandle district of Pennsylvania. Type B.

II. Full retreat with short rib lines. Used by a few mines in Washington County, Pa. Type C.

III. Half advance and half retreat. This practice is common in that part of Allegheny and Washington Counties south of Pittsburgh, Pa. Type D.

IV. Full retreat with long fracture lines. This is the predominant practice in the Westmoreland-Ligonier field and the areas adjacent to the Monongahela River in Fayette, Greene, and Washington Counties in Pennsylvania, and the Fairmont district of West Virginia. Type E.

1.- Pocket-and-stump method of pillar extraction:

(a) Rooms developed on centers of 50 feet or less with correspondingly narrow pillars. Used by certain mines in the vicinity of Greensburg, Pa., with moderately light cover. Group A of type E.

(b) Rooms developed on centers of more than 50 feet with correspondingly thicker room pillars. Used by certain mines in the vicinity of Greensburg, Pa., with moderately heavy cover. Group B of type E.

(c) Rooms developed on block system with rooms on centers of 60 feet or more. Used by mines in Greene, Fayette, and Washington Counties. Group C of type E.

(d) Rooms developed on block system in Fairmont district of West Virginia. Group D of type E.

2.- Open-end system of pillar extraction:

(e) Rooms developed on block system, but pillars are extracted by an open-end method. This is a fairly recent development in the coke district of Pennsylvania. Group E of type E.

This classification shows five types of mining methods based on character of pillar extraction, and type E has been subdivided into five groups. Rather than describe all the variations in these 68 mines, a typical mine for each method will be described and comparisons given with other mines of the same type. The mines described are representative of what is considered to be the best practice in each system. Each of the types could be further subdivided on the basis of minor differences, but as this would not clarify the discussion these differences will be noted in describing each system. The number of mines studied for each type should not be taken as an indication of the order of predominance of each type. More tonnage is produced by type E than all the other types combined. Type A is predominant in eastern Ohio and the Panhandle of West Virginia. Group E of type E is becoming more and more predominant in Fayette County and probably will be used extensively in extracting the large deposits of virgin coal in Greene County, Pa.

Type A.- Full Advance - Room Pillars Not Mined

In the full-advance method the coal is mined on the advance by entry headings and rooms; room pillars, room stumps, and entry-chain pillars are abandoned but barrier pillars are recovered in part. Such a method does not result in a high percentage of extraction, the range being 55 to 65 percent, which means that 35 to 45 percent of the coal is lost. This method of mining is practiced in Belmont, Harrison, and Jefferson Counties in eastern Ohio and in the Panhandle district of West Virginia which includes Brooke, Marshall, and Ohio Counties. The seven mines studied in Ohio and the six in West Virginia are representative of the average practice for this type of development. At the time these mines were studied four had a production of less than 1,000 tons, 3 more than 1,000 tons, 3 more than 2,000 tons, and 3 more than 3,000 but less than 4,000 tons per day.

The average thickness of the Pittsburgh coal bed in these 13 mines is 59 inches. The average thickness at each individual mine does not vary by more than 6 inches from that average except at two mines where the average thickness was 51 to 52 inches; therefore, insofar as thickness of coal is concerned, all 13 mines have about the same condition. Likewise the immediate roof at all mines is similar, consisting, in ascending order, of an average thickness of 11 inches of draw slate which is taken down as the face of the working place advances, 13 inches of roof coal, 7-1/2 feet of calcareous shales and a main roof of limestone. While these are the essential components of the roof, considerable variation is found in the various mines. The draw slate averages about 12 inches in thickness for this district, but there are variations in individual mines from as little as 3 inches to as much as 52 inches of draw slate and these variations may be found on opposite ribs of a heading. The roof coal also varies in thickness but not as greatly as in the case of the draw slate. Figure 2,A, gives some typical coal and roof sections for these mines.

The depth of cover ranges from nothing at outcrop to a maximum of 800 feet. Some of the development features are based on the amount of cover and these mines have been grouped on that basis.

Figure 2,B, represents the plan of development of a mine in Belmont County and is representative of all mines of this type except for certain variations which will be described later. Panels are developed by driving main butt entries and face entries. Butt entries are driven and, as they advance, rooms are driven 24 feet wide on 33- or 34-foot centers. The rooms are driven their full length and then abandoned, no attempt being made to recover the pillar coal or to salvage the timber. Track is laid up the center of the room, and after the room is completed the rails are pulled and the place is then abandoned. Room stumps and butt-entry chain pillars are abandoned when the butt entry is completed. Usually some of the coal in barrier pillars is recovered by driving rooms off face entries in the same manner

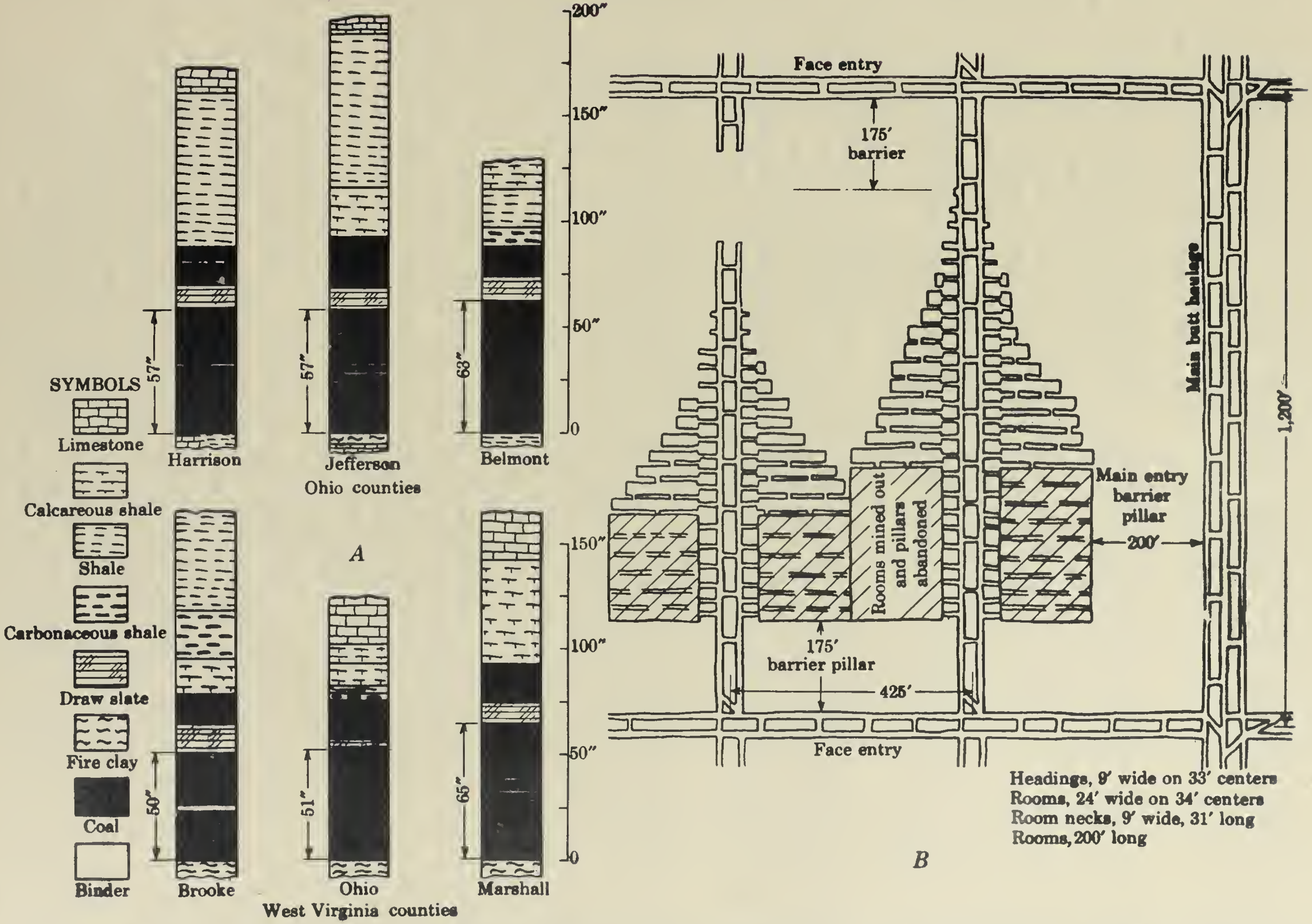

Figure 2.- Coal and roof sections and method of development in the No. 8 field of eastern Ohio and the Panhandle district of West Virginia, type A mines: A, Representative sections of the coal bed and immediate roof in six mines in the counties composing the No. 8 field of eastern Ohio and the Panhandle district of West Virginia; B, method of development of a mine in Belmont County typical of mines in the No. 8 field and the Panhandle district.

that they are driven off butt entries. Naturally, such a method of mining results in a low recovery and it also adds to the cost of haulage, drainage, ventilation, and supervision because to produce a given tonnage the work is of necessity scattered over a large acreage. Less timber should be needed and there should be fewer accidents because pillars are not mined, but this is not necessarily true.

Figure 2,B, gives the plan of development and table 1 gives the details of development and operating results for a representative mine in Ohio and one in West Virginia, and the average for seven mines with average cover of less than 200 feet and for 6 mines with average cover of more than 200 feet. As table 1 shows, the methods of development vary only slightly in each of the 13 mines. The width of pillars is generally increased in those mines having a greater depth of cover. The average length of rooms is about 200 feet. Longer rooms would not be practical because only 8- to 10-foot room pillars are left and a roof squeeze might develop if the rooms had to be maintained for a length of 300 to 400 feet. As the rooms reach the limit of their length, the pillar is gradually reduced until it may be only 2 or 3 feet thick. Rarely are rooms driven on sights, with the result that the thickness of room pillars is liable to be irregular.

Various methods are used to prevent roof squeezes. Barrier pillars, chain pillars and room stumps are used at all mines. In table 1 it should be noted that for light cover the average width of protection on a butt entry, consisting of two room stumps and the butt entry with its two headings and one chain pillar, is 75 feet, and for heavy cover is 88 feet; for light cover 75 percent of this is coal in pillars, and for heavy cover 80 percent is coal in pillars.

In addition to the protection afforded by chain pillars and room stumps other methods of protecting against general squeezes are used. At one mine with an average cover of 500 feet, two rooms off each butt heading near the center of the panel are not mined until all the other rooms on the butt entry panel are completed. This leaves a block of coal 78 feet wide by 200 feet long on each side of the butt entry. Another mine with an average cover of 100 feet drives rooms in groups of five, leaving a solid block of coal 50 feet wide and 200 feet long between each block of five rooms. No attempt is made to recover this coal. A common method used in Ohio for mines with light cover is to leave room blocks as illustrated in figure 3,A. The coal in this room block is seldom recovered. At one mine with an average cover of 500 feet no room blocks were used, but the room headings were driven on 40-foot centers with room necks 17 feet long and squeezes were prevented. Although the usual practice is to drive rooms off both headings as the headings advance, the practice may be varied if the roof is bad; in such places the rooms are completed off one butt heading before those on the parallel butt heading are started. Another variation is to develop the butt entry complete and then drive half of the rooms (on the inby end) before starting the other half of the rooms (on the outby end). At one mine in West Virginia the face entries are turned on 1,200-foot intervals, but the room entries are stopped just short of 600 feet to isolate each butt panel for control of ventilation and for better roof control. On these short 600-foot butt entries all rooms are started at approximately the same time. Another beneficial practice is to drive rooms 200 to 225 feet long off the inby butt heading and those off the outby butt heading only 125 to 145 feet because these are approaching areas which have been mined out, which means about 60 percent recovered, and the roof is riding on the coal that is left in pillars; therefore, it is best that these rooms be short and mined as quickly as possible. The 200 to 225 foot rooms are advancing in solid coal. However, in this case it should be understood that the roof cannot be correctly described as being under control, but is only temporarily supported by pillars. As shown later with the system of mining known as full retreat with long fracture lines, the rooms and pillar pockets are always advancing toward mined-out areas, but the roof is under excellent control.

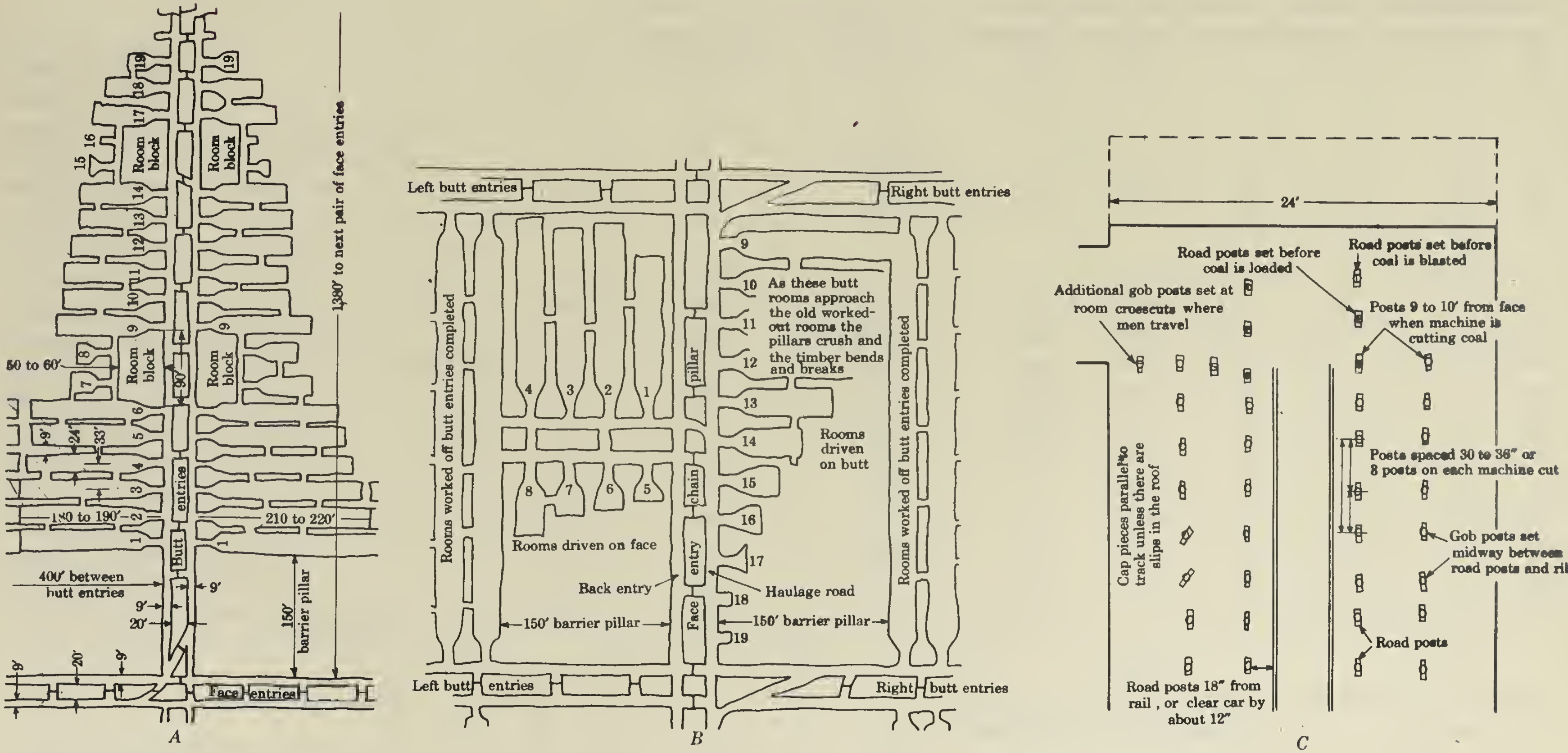

Figure 3.- Details of development and room timbering, type A mines: A, Room blocks left to protect butt entries under light cover in eastern Ohio; B, method of mining part of barrier pillars; C, plan of room timbering in a mine in Belmont County, Ohio.

TABLE 1.- Developmental distances and operating results for 13 mines in No. 8 field of eastern Ohio and the Panhandle district of West Virgina. Type A mines

Developmental distances	Mine in Belmont County (Mine 1)	Mine in Brooke County (Mine 2)	Seven mines, average cover less than 200 feet			Six mines, average cover greater than 200 feet		
			Maximum	Average	Minimum	Maximum	Average	Minimum
Range of cover ... feet	[1]800-500-300	[1]300-175-0	350	121	0	800	400	90
Width of headings ... do.	8 to 9	9	10	9.2	8	10	9.4	8
Number of headings on main entries	3	3 and 4	4	3	2	6	4	2
Number of headings on face entries	2	3 and 4	4	2	2	4	2	2
Heading centers on mains and faces ... feet	32	30	40	32	29	80	45	33
Heading centers on butts ... do.	32	30	40	33	29	50	39	30
Thickness of chain pillars on mains and faces ... do.	23.5	21	30	23	20	70	36	21
Thickness of chain pillars on butts ... do.	23.5	21	30	23	20	41	29	21
Distance between face entries ... do.	1,200	1,300	1,950	1,350	1,050	1,400	1,175	1,000
Center to center distances of butt entries ... do.	425	380	490	433	380	450	421	[2]350
Barrier pillars on mains ... do.	200	200	200	162	80	450	251	200
Barrier pillars on faces ... do.	175	100	200	141	80	200	173	150
Width of rooms ... do.	24	24	26	24	24	24	24	22
Width of room pillars ... do.	10	8	9	8	6	11	8.6	6
Room centers ... do.	34	32	33	33	32	35	33	30
Length of room ... do.	200	135 and 215	250	198	125	240	206	155
Length of room necks[3] ... do.	31	20	20	18	15	31	17	10
Width of protection on butt entry[4] ... do.	102.5	79	83	75	71	102.5	88	74

Operating results	Mine 1	Mine 2	Twelve hand-loading mines			Mechanical-loading mine
			Maximum	Average	Minimum	
Coal recovery ... percent	62.3	65	65	57	55	55
Coal produced per day per man underground ... tons	6.2	7.3	...	5.4	...	7.6
Coal produced per post used ... do.	[5]4.4	3.95	...	4.68	...	4.63
Coal produced per pound of explosive ... do.	(6)	10.5	...	7.81	...	4.88
Fatality rate per million tons of coal produced	0.91	3.60	...	4.81	...	3.21
Frequency rate for all injuries underground	179.7	25.96	...	122.49	...	130.2
Severity rate for all injuries underground	9.2	25.86	...	23.68	...	19.8

[1]Three values represent maximum, average, and minimum. [2]Does not include mines that turn rooms off one butt heading only.
[3]Width of room necks is same as width of headings. [4]Consists of width of chain pillar, two headings, and two room stumps.
[5]Estimated. [6]Not available.

At a mine in Belmont County, Ohio, with 450 to 630 feet of cover, the rooms are turned off only one butt entry, thus the rooms are always advancing toward solid coal. Butt headings are not driven through to the next face entry and rooms are stopped 20 to 30 feet short of the next butt heading. Butt entries are developed in groups of four and the rooms on the outby entry of this group are not started until the rooms on the three pairs of inby butt entries are completed. The whole butt entry panel, 260 by 1,000 feet, thus acts as a safety block to prevent squeezes. On each butt entry half of the rooms (on the inby end) are driven 150 feet before the other half start, then all rooms are driven simultaneously.

A mine with an average cover of 400 feet in Marshall County, W. Va., was obtaining successful results by driving rooms 60 feet wide. Rooms are turned on 36-foot centers, but the room pillar of a pair of rooms is mined out after leaving a 40-foot room stump. A track is laid up each room neck and as a result two tracks are on each side of the 60-foot room. With rooms 24 feet wide, considerable trouble was experienced with roof falls in rooms, but with 60-foot rooms this trouble has been eliminated.

An attempt is made to recover some of the coal in the barrier pillars. The details of work vary but the general method of attack is shown in figure 3,B. It consists simply of driving rooms into the barrier pillars. In one section of a mine in Marshall County, W. Va., some room-pillar recovery was attempted. Here the rooms are turned off only one butt heading. The rooms are driven 155 feet long, 20 feet wide, and with a 14-foot pillar which is slabbed by two machine cuts after the room is driven. Other attempts have been made in this field to recover room pillars, but no systematic scheme, such as will be described subsequently, has ever been given a fair trial.

Coal is undercut by mining machines and in all mines 100 percent of the production is machine mined. The general practice is to work one miner to a place, or two men to two places This lack of concentration of workers results in a low daily production per room and, therefore, extensive development is required for a given daily mine tonnage.

Figure 3,C, illustrates an excellent system of room timbering for this type of mining. Table 1, together with the illustrations and the foregoing discussion, describe this system of mining. Critical comments on this method will be given after the other systems have been discussed.

Type B.- Full Advance with Pillar Extraction

The next step in a mining method is full advance with pillar extraction. This method is similar to Type A in the development of panels, but differs in that an attempt is made to recover all or part of room pillars, room stumps, and chain pillars, so that a greater percentage of recovery is attained. Type B represents a modification of type A which adapts it to use in coal lands of greater value from which a higher recovery is desired. Development methods are similar to those of type A except that thicker room pillars are left as the rooms are driven. The rooms are developed as the butt entry is driven, but as soon as the room reaches its limit, the extraction of the room pillar is begun. In some instances poor management allows rooms to stand for some time before an attempt is made to extract the pillars. Successive room pillars are mined in step back to the room stump. Rooms are turned off both butt headings. Thus on a butt entry the work may consist of driving the headings, advancing the rooms, and extracting the room pillars. The percentage of recovery is probably between 70 and 85 percent for this type of mining, but no accurate values were obtained.

Four mines of this type were studied in the Panhandle district of Pennsylvania, but they are representative of many mines using this type of development and pillar extraction. Two of the mines produced less than 1,000 tons per day and two more than 1,000 but less than 2,000 tons per day.

The average thickness of the Pittsburgh coal bed in these four mines is 59 inches and varies locally from 56 to 63 inches. The draw slate varies in thickness from 4 to 30 inches, but the general average for all the mines is 13 inches; however, the average at individual mines varies from 9 to 22 inches. Above the draw slate, which is taken down as the face of the working place advances, is the usual roof coal with an average thickness of 9-3/4 inches; between the roof coal and the main roof are 9 to 15 feet of shales and thin laminated beds of coal. The main roof varies from a sandstone to a sandy shale and is 10 to 30 feet thick. Figure 4,A,gives some representative sections of the coal bed and the immediate roof.

In the area where this system is used the cover is light and it should be noted in Table 2 that the maximum cover is only 300 feet and the average cover is 95 feet. In three of the mines the development is largely controlled by outcrop lines.

TABLE 2.- Developmental distances and operating results for four mines in the Panhandle district of Pennsylvania. Type B mines

Developmental distances		Representative mine (Mine 3)	Range for four mines Max.	Avg.	Min.
Range of cover	feet	[1]250-125-65	300	95	5
Width of headings	do.	10 to 11	11	10.2	10
Number of headings on main entries		4	4	3	3
Number of headings on face entries		2, 3 and 4	4	3	2
Heading centers on mains, faces and butts	feet	40 and 50	50	44	40
Thickness of chain pillars on mains, faces and butts	do.	30 and 40	40	34	30
Distance between face entries	do.	1500	1800	1267	1000
Center to center distances of butt entries	do.	495	[2]650	[2]516	[2]450
Barrier pillars on mains	do.	40	125	72	40
Barrier pillars on faces	do.	20 and 40	125	51	20
Width of rooms	do.	24	24	23	21
Width of room pillars	do.	13	13	11	9
Room centers	do.	37	37	34	30
Length of rooms	do.	225	300	238	200
Length of room necks[3]	do.	21	21	19	18
Width of protection on butt entries[4]	do.	98	98	91	86
Width of pillar pocket	do.	20 to 24	24	17	12
Thickness of coal fender for pillar pocket	do.	3 to 4	8	4	1
Operating results					
Recovery	percent	80		[5]82.5	
Coal produced per day per man underground	tons	4.4		4.9	
Coal produced per post used	do.	4.78		[6]5.05	
Coal produced per pound of explosive	do.	6.8		[5]6.9	
Fatality rate per million tons of coal produced		2.38		0.60	
Frequency rate for all injuries underground		122.99		157.26	
Severity rate for all injuries underground		13.58		9.20	

[1]Three values represent maximum, average, and minimum values.
[2]Does not include mine driving rooms off one butt heading only.
[3]Width of room necks is same as width of headings.
[4]Consists of width of chain pillars, two headings, and two room stumps.
[5]For two mines only.
[6]For three mines only.

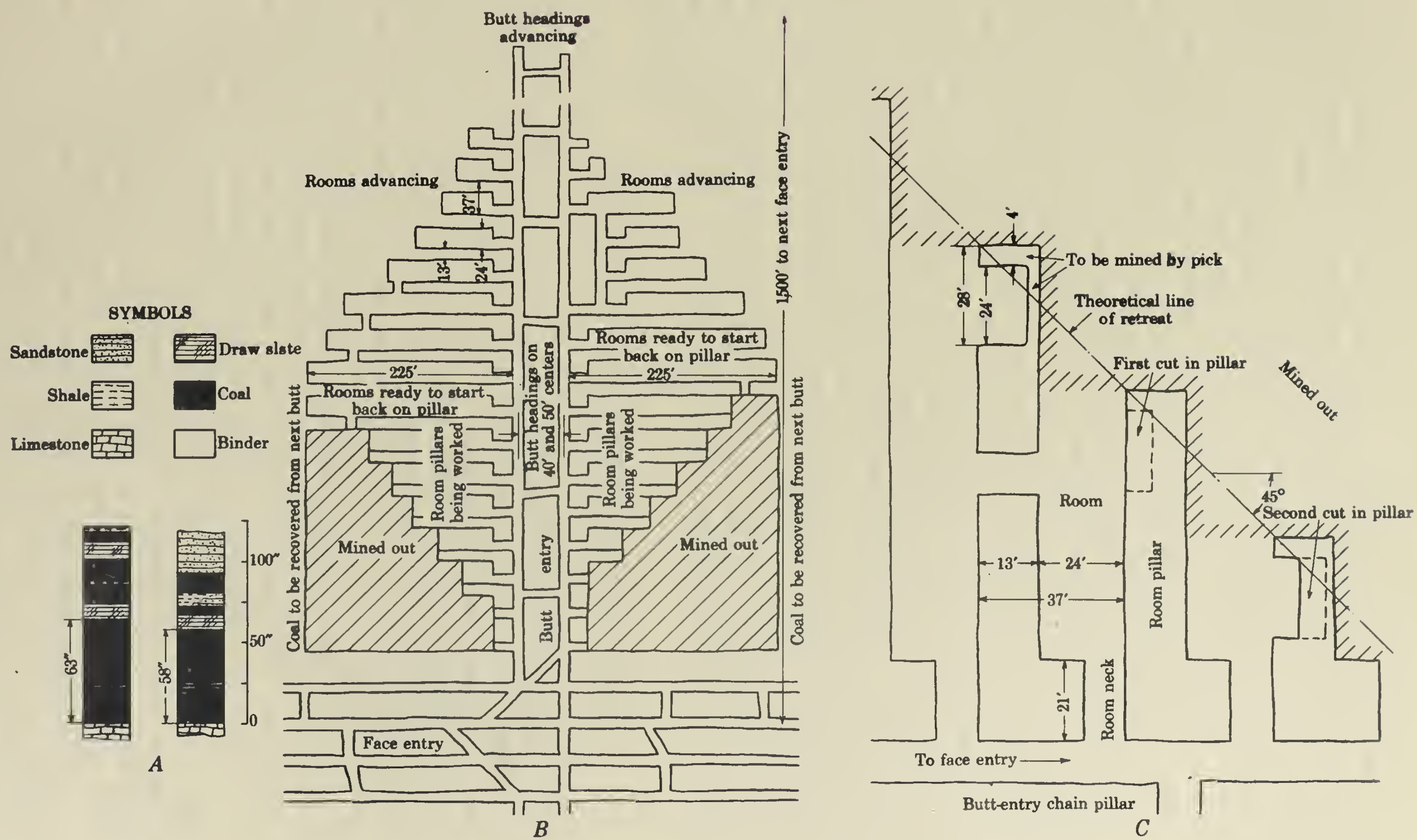

Figure 4.- Coal and roof sections, plan of development, and pillar extraction used in the Panhandle district of Pennsylvania, type B mines: A, Sections of the coal bed and immediate roof typical of type B mines; B, method of development and pillar extraction on a butt entry; C, details of extraction of room pillars.

Figure 4,B, shows the general plan of development and pillar extraction. Table 2 gives the details of development distances and operating results. The method is quite similar to type A except that rooms are a few feet narrower and room pillars are a few feet thicker; the room centers, however, are practically the same. Rooms are driven on sights at only one mine. The difference between the two types is that room pillars are extracted and some attempt is made to recover room stumps, chain pillars, and barrier pillars. Since the cover is light, the necessity for maintaining uniformly retreating pillar lines is not so essential as is the case where moderate to heavy cover is encountered. Figure 4,C, shows the standard method of pillar extraction. The general practice is to keep the pillars of 3 to 7 rooms retreating on an angle of approximately 45° with respect to the butt entry. With heavy cover many of the practices permissible at these mines would result in serious consequences.

As in type A, the development of squeezes along butt entries must be prevented. Chain pillars and room stumps provide this protection, at least while the rooms are being driven and the room pillars extracted. The total width of the two room stumps and the butt entry with its two headings and one chain pillar averages 91 feet and 71 feet, or 78 percent of this is solid coal; and this is for an average cover of only 95 feet. The methods of development for all four mines is well represented by figure 4,B; however, in one section of one of the mines, rooms were turned off only one butt heading. After the rooms and room pillars are extracted on a butt-entry panel, then the chain pillars and room stumps are removed, beginning at the inby end of the butt. Figure 5,A, shows the practice in mining chain pillars and room stumps at one of the mines; figure 5,B, shows the best practice for room timbering; and figure 5,C, shows details of pillar work and timbering in pillar places.

Mining machines are used to undercut the coal whenever possible, but that is not always practicable in pillar work. From 75 to 99 percent of the production at the individual mines is machine mined.

Type C.- Full Retreat with Short Fracture Lines

Full retreat with short fracture lines is a method of development and pillar extraction far superior to that used in type B, because it permits better roof control and concentration of work. In this type no rooms are driven or room pillars extracted until the butt entry is fully developed, then rooms are driven and pillars extracted. Room necks may be turned as the butts are developed. Room stumps and chain pillars are extracted in step with room pillars as the work retreats back to the face entries. The percentage of recovery is probably between 80 and 85 percent and could be higher except that often small pillar stumps are abandoned because their recovery would be hazardous. Only three mines using this system were studied: One in the northwestern part of Washington County, Pa., and the other two in the southeastern part of the same county. One of these mines produced 1,000 tons daily and the other two each produced about 5,200 tons daily. Since only three mines of this type were studied, the data with respect to each one will be given separately in table 3. These mines have been designated respectively as Nos. 4, 5, and 6.

At mine 4 the coal is 58 inches thick and at mines 5 and 6 the coal ranges in thickness from 66 to 69 inches. Figure 6,A, gives typical sections of the coal bed and immediate roof at these mines. As shown in table 3, it is well to bear in mind when considering the differences in development between these three mines that mine 4 has an average cover of 175 feet and mines 5 and 6 have an average cover of 500 and 600 feet, respectively.

Figure 6,B, gives the plan of development for mines 5 and 6. Development is similar at mine 4 except that its rooms are driven on 34-foot centers as compared to 45- and 90-foot centers for mines 5 and 6. Butt entries are fully developed before any rooms are turned except that the first two rooms off each butt heading are driven to facilitate ventilation.

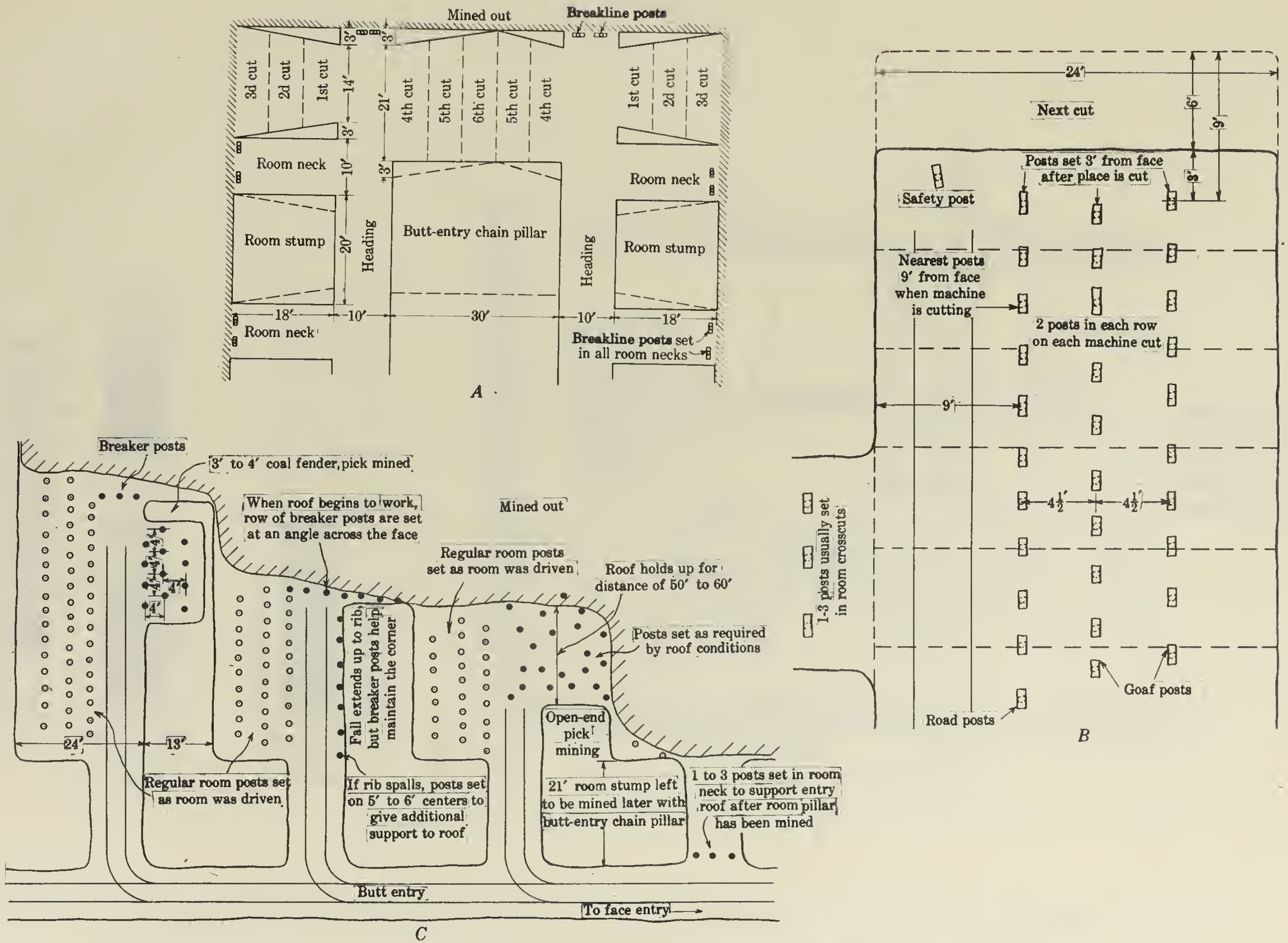

Figure 5.- Details of pillar extraction, room timbering, and timbering in pillar work, type B mines: A, Plan of recovering butt-entry chain pillars and room stumps; B, plan of room timbering; C, plan of rooms showing methods of pillar extraction and timbering in pillar places.

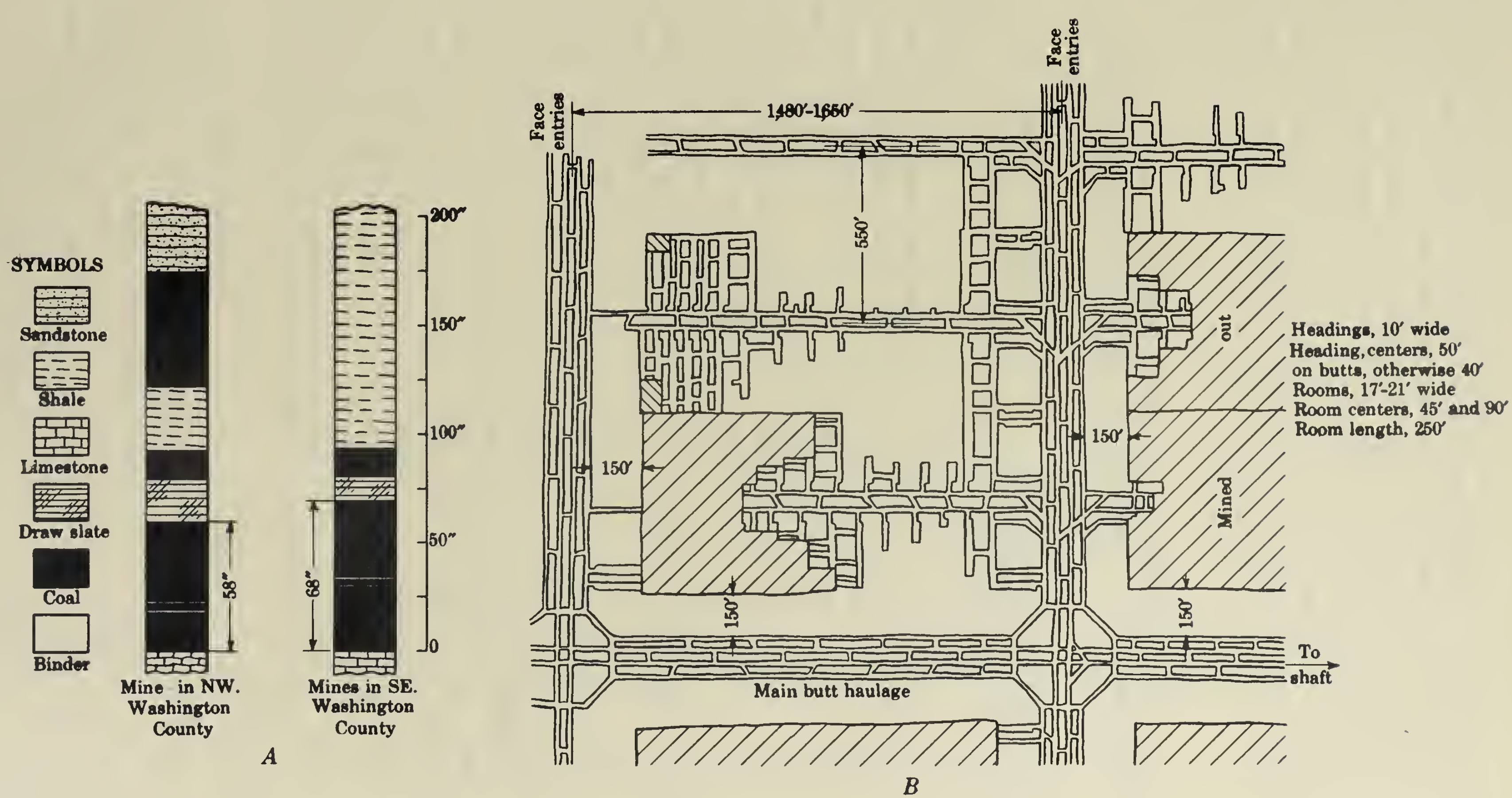

Figure 6.- Coal and roof sections and method of development when using full retreat with short pillaring lines, type C mines: A, Representative sections of the coal bed and immediate roof; B, plan of development and pillar extraction for a mine in the southeastern part of Washington County.

After the butt entry is fully developed, rooms are started at the inby end and then, as soon as each room reaches its limit, pillar extraction begins. Room stumps and chain pillars are extracted in step with the room pillars. At mine 4 the extraction of chain pillars sometimes lags behind the extraction of room pillars. With light cover, this may not entail serious consequences, but the best practice is to maintain an even fracture line across rooms and chain pillars. This practice is carried out at mines 5 and 6 where serious trouble would develop because of the greater depth of cover if the extraction of chain pillars was allowed to lag behind the regular retreat of room pillars. At mine 4 the fracture line extends at an angle of 45° across three or four rooms, but at mines 5 and 6 the angle of retreat is at 27° 40' across four rooms.

TABLE 3.- Developmental distances and operating results for three mines in Washington County, Pa., using full retreat with short pillaring lines. Type C mines

Developmental distances	Northwestern part of county	Southeastern part of county	
	Mine 4	Mine 5	Mine 6
Range of cover[1]....feet	240-175-90	900-500-250	845-600-355
Width of headings....do.	10	10	10
Number of headings on main entries....	5	4	6
Number of headings on face entries....	3	3 and 4	2 to 6
Heading centers on mains....feet	50	40	40
Heading centers on face....do.	50	40	40 and 50
Heading centers on butts....do.	50	50	50
Thickness of chain pillars on mains....do.	40	30	30
Thickness of chain pillars on faces....do.	40	30	30 and 40
Thickness of chain pillars on butts....do.	40	40	40
Distance between face entries....do.	1,500 to 1,800	1,480 to 1,650	1,400
Center to center distances of butt entries do.	450 to 550	550	550
Barrier pillars on mains....do.	150	150	150
Barrier pillars on faces....do.	100	150	150
Width of rooms....do.	21	17 to 21	17 to 21
Width of room pillars....do.	13	25 and 70	25 and 70
Room centers....do.	34	45 and 90	45 and 90
Length of rooms....do.	200 to 250	250	250
Length of room necks....do.	18	25 to 30	30
Width of primary pillar pocket....do.	20	17	17
Width of primary pillar fender....do.	6	11	11
Width of secondary pillar pocket....do.	....	21	21
Width of secondary pillar fender....do.	....	9	9

Operating results	Mine 4	Average for mines 5 and 6	Average for all three mines
Recovery....percent	85	85	85
Coal produced per day per man underground....tons	4.63	7.89	7.44
Coal produced per post used....do.	2.99	5.55	7.44
Coal produced per pound of explosives....do.	Information not available		
Fatality rate per million tons of coal produced	1.77	1.86	1.83
Frequency rate for all injuries underground....	99.03	66.20	77.14
Severity rate for all injuries underground....	9.96	10.76	10.49

[1]Three values represent maximum, average, and minimum values.

Because of the lighter cover, rooms are turned on 34-foot centers at mine 4, but on 90-foot centers at mines 5 and 6; however, the first six inby rooms at the latter two mines are turned on 45-foot centers. This is done to speed up the initial production from the butt entry. At mines 5 and 6 the room width may vary from 17 to 21 feet. This is because the clean-up system is used and the places are cut each day according to the ability of the miners to load coal. All rooms are driven on sights. In figure 6,B, note the method of development used at the intersection of main entries to avoid numerous small chain-pillar blocks.

Figure 7 illustrates details of pillar extraction and timbering in rooms and pillar places.

Bottom cutting machines are used at all three mines. At mine 4, 95 percent of the coal is machine mined; at mines 5 and 6 an average of 70 percent of the coal is machine mined.

Type D.- Half Advance and Half Retreat

The half-advance and half-retreat system is so called because as the butt entries are driven the rooms are turned off one heading and, after the rooms reach their limit the room pillar is mined. After the butt entry has reached its limit the rooms off the other heading are driven, starting at the inby end of the butt entry. The pillars of these rooms and the chain pillar and room stumps of both headings are then mined as the work retreats to the face entry. The rooms on the advance side are driven 50 to 90 feet longer than those on the retreat side. This is done because on the advance there is considerable solid coal for protection against squeezes; whereas on the retreat work the pillar extraction can be done quicker by having shorter rooms. The recovery varies from 72 to 90 percent, with an average of 82 percent for six of the mines.

This method of development is used principally in those parts of Allegheny and Washington Counties south of Pittsburgh. Seven mines using this system were studied. Six of the mines produced 1,500 to 2,000 tons daily and the seventh about 3,100 tons daily.

The coal bed varies in thickness from 56 to 93 inches. The average thickness for six of the mines is 62 inches and for the seventh mine is 93 inches. The immediate roof consists of the usual draw slate and roof coal and the main roof is generally a hard shale. The draw slate averages 12 inches in thickness for the seven mines. The general range in thickness of draw slate at individual mines is 5 to 15 inches; however, at two of the mines a maximum of 3 and 6 feet of draw slate was observed. Beds of coal and shale are found between the draw slate and the main shale roof. The total thickness varies from 10 inches to 4-1/2 feet. The thickness of the roof coal directly over the draw slate varies from 2 to 22 inches and averages 11 inches. The average cover varies from 125 to 400 feet. Figure 9,A, shows typical coal and roof sections.

Figure 8 shows the general plan of development and table 4 gives the details of development, pillar extraction, and operating data. Two typical mines are selected for this group.

Roof control is obtained by leaving rather substantial chain pillars and room stumps on the advance; one mine has a chain pillar 54 feet wide. The critical point, insofar as roof control is concerned, comes when the change is made from advance to retreat on the individual panel. Too often extraction of the chain pillar and room stumps lags behind extraction of room pillars, with the result that these stumps extend into the gob, start squeezes, and much coal is lost. Another mistake is to develop too many rooms ahead of the pillar work so that rooms must stand some time before their pillars are mined. When used with moderate cover, close attention to limiting the solid development, and rapid extraction of pillars in proper step, this system works satisfactorily. Fracture lines are short and generally extend across four to seven rooms, depending upon the length of the rooms, because the pillars retreat in steps of 25 to 50 feet.

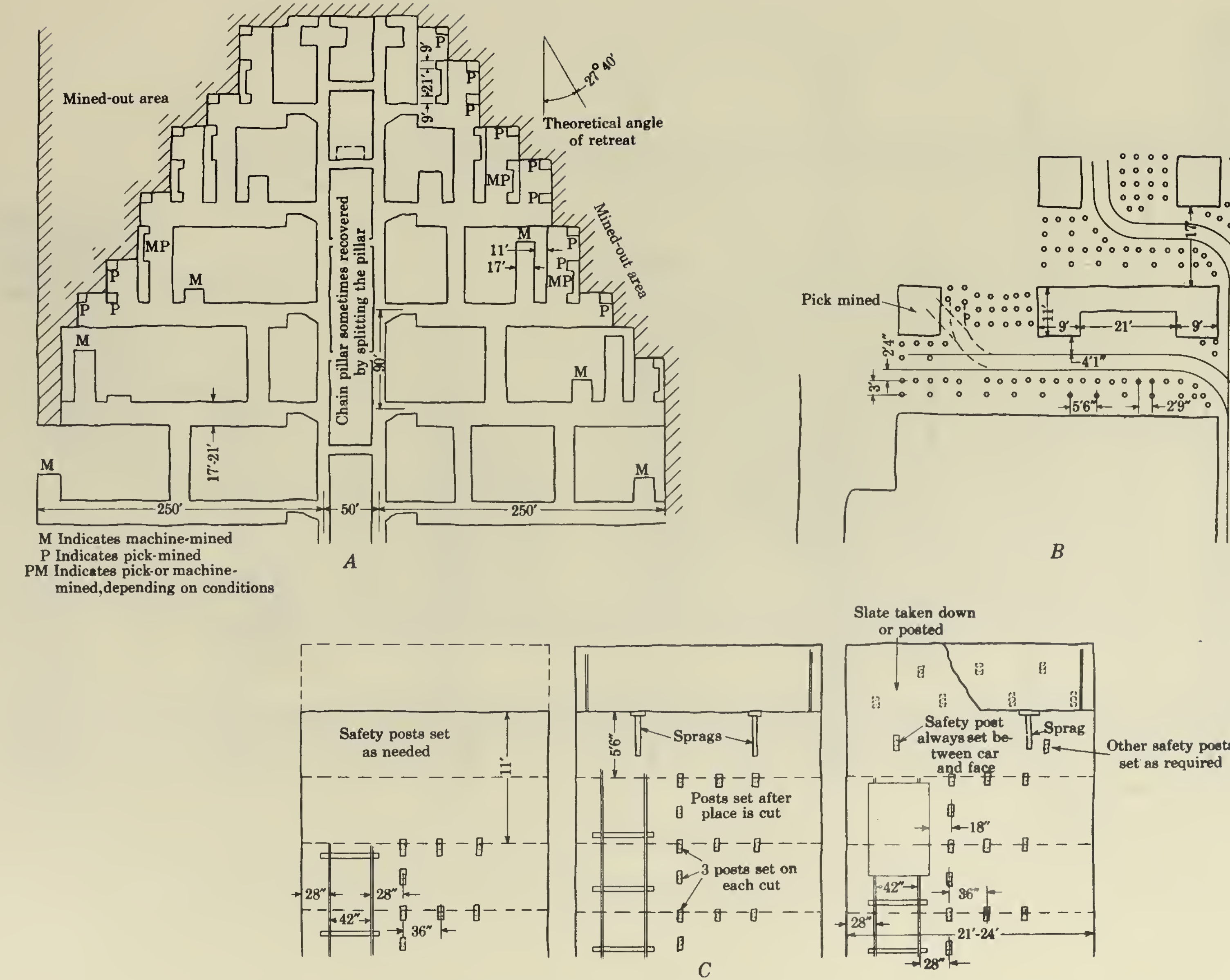

Figure 7.- Details of pillar extraction and timbering, type C mines: A, Details of pillar extraction on butt entry; B, details of timbering in pillar work; C, method of timbering in rooms.

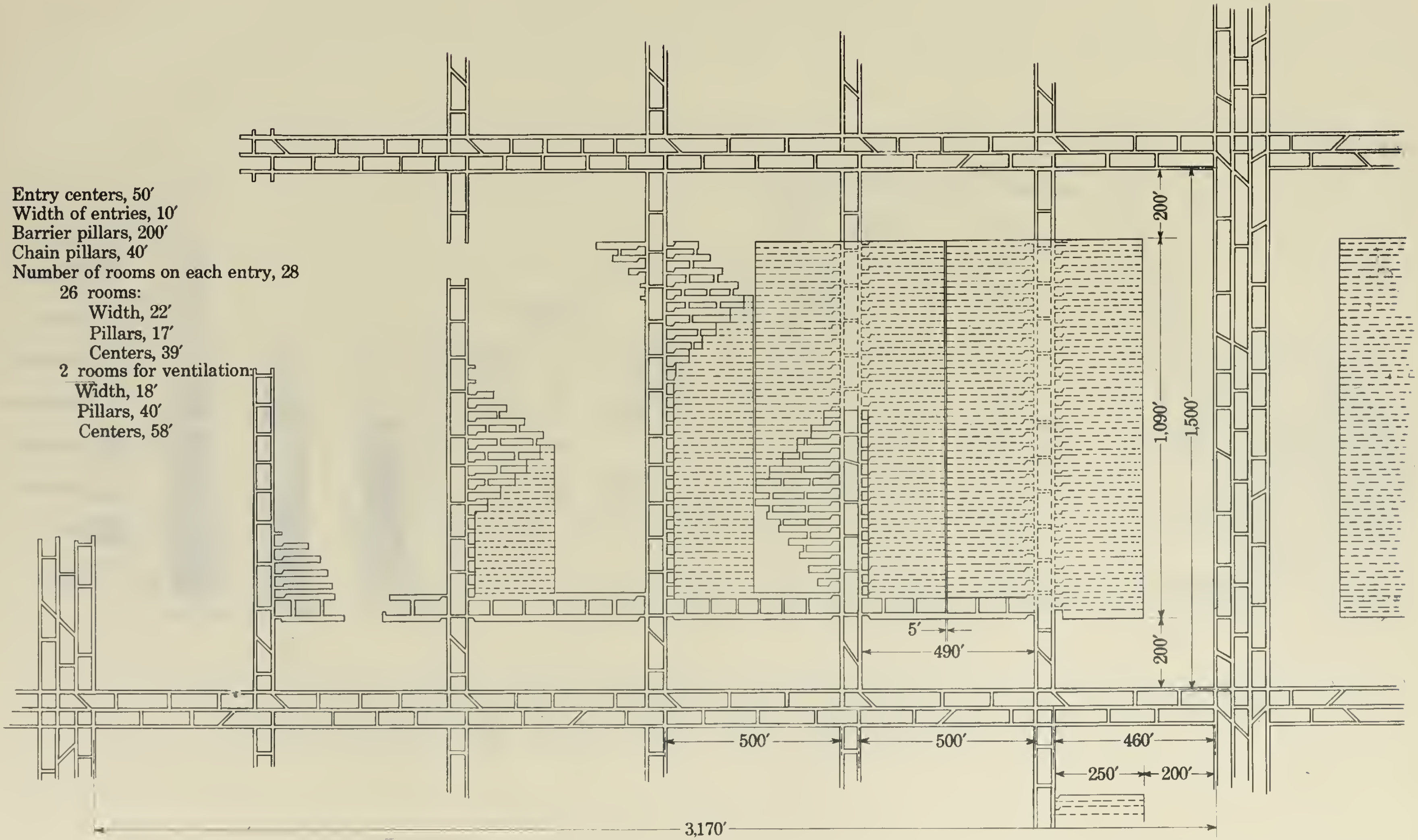

Figure 8.- Method of development of mines using the half-advance and half-retreat system of pillar extraction, type D mines.

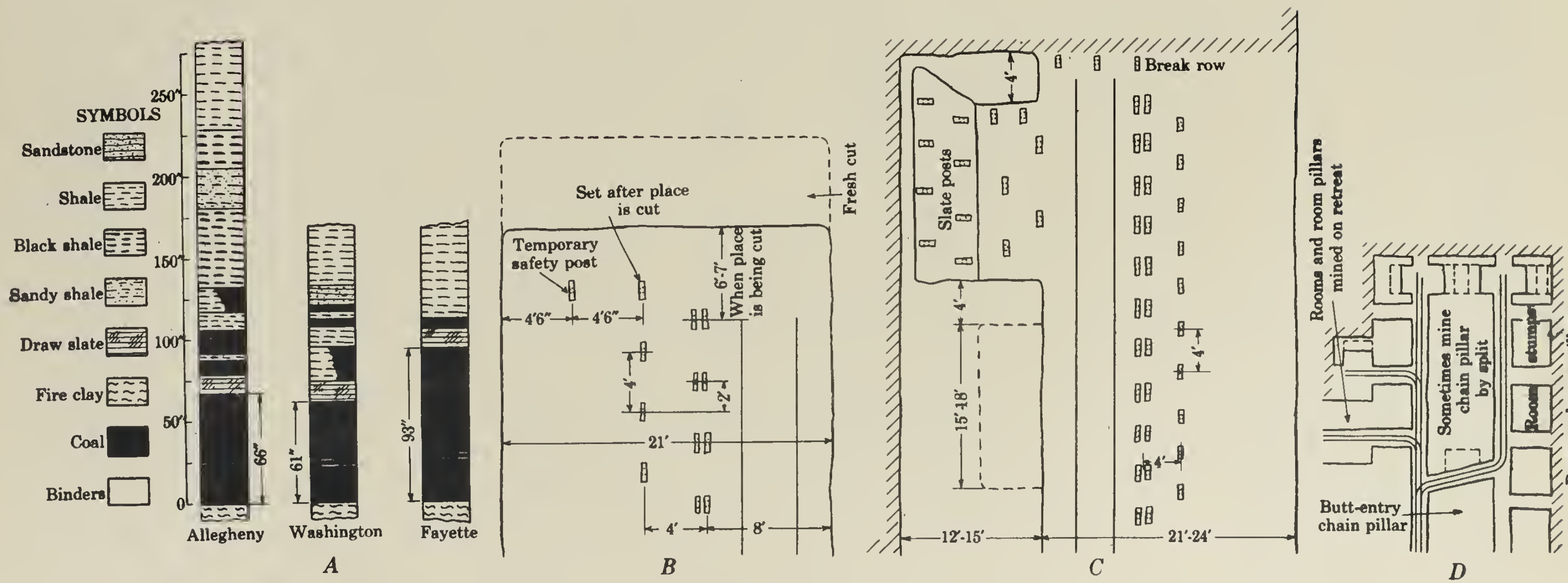

Figure 9.- Details of coal and roof sections, timbering, and pillar extraction, type D mines: A, Representative sections of coal bed and immediate roof of mines in group D in Allegheny, Fayette, and Washington Counties, Pa.; B, plan of room timbering; C, timbering in pillar pockets; D, method of extracting chain pillars and room stumps.

TABLE 4.- Developmental distances and operating results for mines using the half-advance and half-retreat system in Allegheny and Washington Counties, Pennsylvania. Type D mines

Developmental distances	Mine in Allegheny County (Mine 7)	Mine in Washington County (Mine 8)	Range for seven mines Maximum	Average	Minimum
Range of cover[1] ... feet	400-300-200	400-250-25	600	284	25
Width of headings ... do.	10	11	11	10.6	10
Number of headings on main entries ...	4	4	4	4	3
Number of headings on face entries ...	3 or 4	2 or 4	4	3	2
Heading centers on mains and faces ... feet	50	40	50	47	40
Heading centers on butts ... do.	50	40	65	49	40
Thickness of chain pillars on mains and faces ... do.	40	29	40	37	29
Thickness of chain pillars on butts ... do.	40	29	54	38	29
Distance between face entries ... do.	1,400 to 1,950	1,450	1,950	1,539	1,200
Center to center distances of butt entries ... do.	480	550	550	496	435
Barrier pillars on mains ... do.	200	100	300	146	90
Barrier pillars on faces ... do.	200	100	200	120	90
Width of rooms ... do.	21	24	24	22.3	18
Width of room pillars ... do.	18	15	27	16	12
Room centers ... do.	39	39	45	39	36
Length of rooms, on advance[2] ... do.	275	280	350	281	270
Length of rooms, on retreat[3] ... do.	200	200	220	181	100
Length of room neck[4] ... do.	24	21	30	23	18
Width of pillar pocket ... do.	17	19	22	17.5	15
Width of pillar fender ... do.	6	4	6	5	4
Operating results					
Recovery ... percent	86	90		[5]82.0	
Coal produced per day per man underground ... tons	4.80	3.03		3.89	
Coal produced per post used ... do.	4.81	5.51		5.63	
Coal produced per pound of explosives used ... do.	11.3	12.3		[6]9.5	
Fatality rate per million tons of coal produced ...	4.16	4.11		3.98	
Frequency rate for all injuries underground ...	79.33	135.82		142.02	
Severity rate for all injuries underground ...	20.67	12.10		18.81	

[1]Three values represent maximum, average, and minimum values.
[2]Rooms driven off one heading as butt entry advances.
[3]Rooms driven off other heading as butt entry work is retreating.
[4]Room necks same width as entries.
[5]For six mines only.
[6]For three mines only.

The method of timbering illustrated in figure 5,B, is used at one of these mines. Figure 9 shows further details of pillar extraction and timbering. From 83 to 97 percent of the production is undercut by machine.

Type E - Full Retreat with Long Fracture Lines

In the method of mining by full retreat with long fracture lines the area to be mined is first developed into panels by main-butt haulage roads and face entries. Panel sizes vary and depend upon many factors, but an average panel would be 1,200 to 1,600 feet wide by 1,800 to 2,400 feet long. Butt entries are driven off the face entries, starting at one end of the panel, but no rooms are turned until the butt entry is completed. Rooms are turned off one butt heading only, and as soon as the room is completed, the extraction of the room pillar begins. The rooms are driven in step and timed so that as soon as they are completed their pillars can be extracted in step with a general rib line. The pillars are mined in step with the rib line, making an angle of 45° or less with the butt entries. The length of the rib line depends upon the size of the panel. Room stumps and chain pillars are mined in step with the general rib line.

Forty-one mines studied in western Pennsylvania and the Fairmont district of West Virginia were using this system of mining. With the variable natural conditions encountered over this large area it is to be expected that the methods used should differ in detail. A subclassification of type E can be made on the basis of size of room pillars, as follows:

1.- Pillars developed on the block system, or, rooms on 50- to 125-foot centers.

2.- Medium room pillars developed, or, rooms on 33- to 50-foot centers.

Another classification is based on the method of mining pillars by (1) the customary pocket-and-stump method or by (2) the open-end system. Still another classification is possible on the basis of whether pillars are mined by butt pockets only or by alternate butt and face pockets.

To describe these 41 mines they will be grouped primarily on a geographical basis which to some extent is also a classification according to the details of mining.

Figure 10,A, shows the plan of development for some mines in Washington and Westmoreland Counties, Pa., with rooms on centers of 50 feet or less and with room pillars 13 to 29 feet wide. These are discussed under group A of type E (See p.15). Figure 10,B, shows the plan of development for mines in which the rooms are driven on centers of 50 to 80 feet with room pillars 40 to 60 feet wide. This method is used in Westmoreland, Fayette, Washington, and Greene Counties, Pa., and in the Fairmont field of West Virginia. Mines of this type are described under groups B, C, and D of type E. Figure 11,A, represents the block system of development used by mines in the Fairmont district of West Virginia and the coking district of Pennsylvania and are described under groups C, D, and E of type E. Figure 11,B, represents a type of development used by some mines in Pennsylvania and West Virginia in which chain pillars for all entries are developed in blocks of the same size as room-pillar blocks. Note that the angle of retreat is less than 45° with respect to the butt entries. Figures 10 and 11 are taken from maps of individual mines and therefore represent special conditions; however, they are representative of all mines in type E. For open-end mining, the type of development may vary but would follow some plan such as shown in figures 10,B, 11,A, and 11,B. As seen from these illustrations, the general plan of development by main-butt haulage roads and by face and butt entries does not vary very much. The principal variation occurs in the method of pillar recovery. These details will be discussed as each group of type E is described.

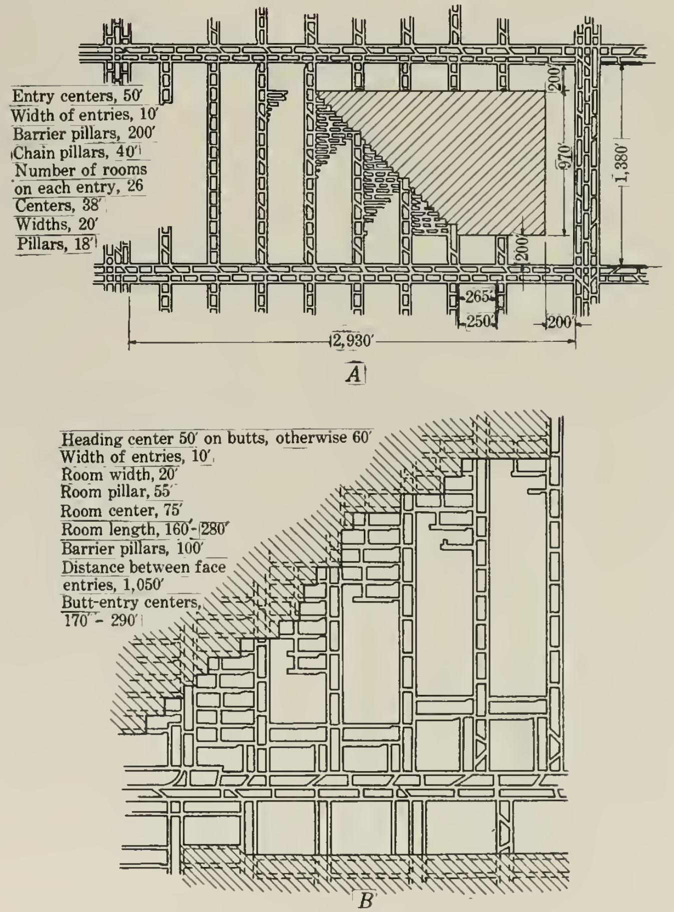

Figure 10.- General plans of development, type E mines: A, Plan of development and pillar extraction for mines driving rooms on centers of 50 feet or less, typical of mines in group A; B, plan of development and pillar extraction for mines driving rooms on centers of 50 feet or more, typical of mines in groups B, C, and D.

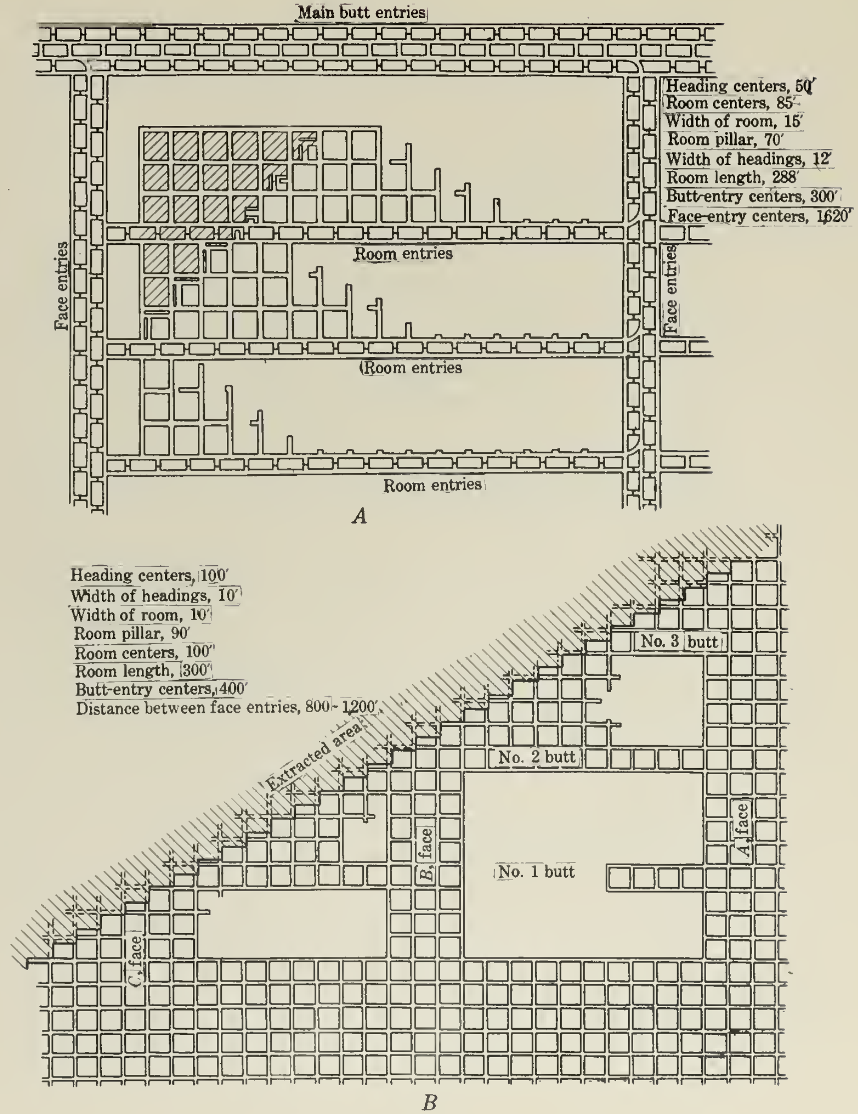

Figure 11.- General plan of development, type E mines: A, Development and pillar extraction using the block system typical of mines in the coking district of Pennsylvania and the Fairmont district of West Virginia; B, development by driving all solid work narrow and on 100- to 125-foot centers, typical of some mines in groups B, C, D, and E.

Group A of Type E

Group A consists of six mines, four in Westmoreland County near Greensburg and two in the northern part of Washington County. This group is distinguished from others of the same type by the fact that rooms are developed on centers of 50 feet or less and are driven 18 to 21 feet wide with pillars 13 to 29 feet wide. Recovery ranges from 85 to 90 percent. Two of the mines produce less than 1,000 tons, two produce between 1,000 and 2,000 tons, one produces 2,500 tons, and one 3,600 tons per day.

The Pittsburgh coal bed in the four mines in Westmoreland County averages 82 inches in thickness. The draw slate varies in thickness from 6 to 12 inches, with an average of 9 inches. The roof coal averages 15 inches in thickness and varies from 9 to 28 inches but, when attaining its maximum thickness, it usually contains many thin beds of shale. Between the roof coal and the main roof, which varies from a shale to a sandstone, are alternate beds of coal and shale having a total thickness of 30 to 55 inches. In the two mines in Washington County the coal and immediate roof is rather uniform, consisting, in ascending order, of 65 inches of coal, 10 inches of draw slate, 16 inches roof coal, 33 inches of shale and coal, and a main roof of shale. Figure 12,A, shows representative sections of the coal bed and roof for these mines in group A and for mines in group B which also are in Westmoreland County. In general, the coal bed is flat, but in a few of the mines in Westmoreland County there are local pitches where the coal reaches a maximum dip of 5 to 6 percent. The average cover at the individual mines ranges from 65 to 400 feet, and it should be noted that the development and extraction of thin pillars is best suited for light to moderate cover.

Figure 10,A, shows the general plan of development and the extraction of the pillars along definitely planned rib lines. Table 5 gives the development distances and operating results of this group of mines.

Because of the moderate cover, roof control is not difficult in these mines. At one mine having a maximum cover of 550 feet the butt entries are on 350-foot centers, but a split panel entry or subbutt heading is driven so that short rooms can be used and the pillars recovered immediately. (See figure 12,B.) When the rooms were driven 300 feet in length considerable trouble was experienced with the roof in the rooms before the room pillar was extracted. By driving shorter rooms this trouble has been eliminated. At this same mine the pillars are mined open end by pick work, but this is not the same open-end system of pillar extraction as used in the coking region where the open-end pillar places are cut by machine. Figure 12,C, shows the method of timbering in pillar places for this same mine. Another mine with a maximum cover of only 100 feet drives 10-foot rooms on 100-foot centers as the butt entry is driven, thus obtaining greater initial production as the butt develops. fter the butt entry is completed and pillar work begins, two additional rooms are driven in the pillars, thus reducing the centers to 33-1/3 feet. This system is possible with light cover and with an immediate roof which can be supported for a definite period of time without the necessity for retimbering. Figure 13,A, shows the details of pillar work. Note that the rooms advance in steps of 25 feet and the pillars retreat in steps of 50 feet. Figures 13,B and C, show the methods of timbering in rooms and pillar places for this same mine.

Coal is undercut by machines at all mines and varies from 51 to 90 percent of the production. The pillars are recovered by driving pockets by machine and leaving thin stumps or fenders which are mined by machine if possible and otherwise by pick. Pillar lines retreat at an angle of 45° with respect to butt entries, although in one mine the angle is 52° and at another the angle is 24° with respect to the butt entry.

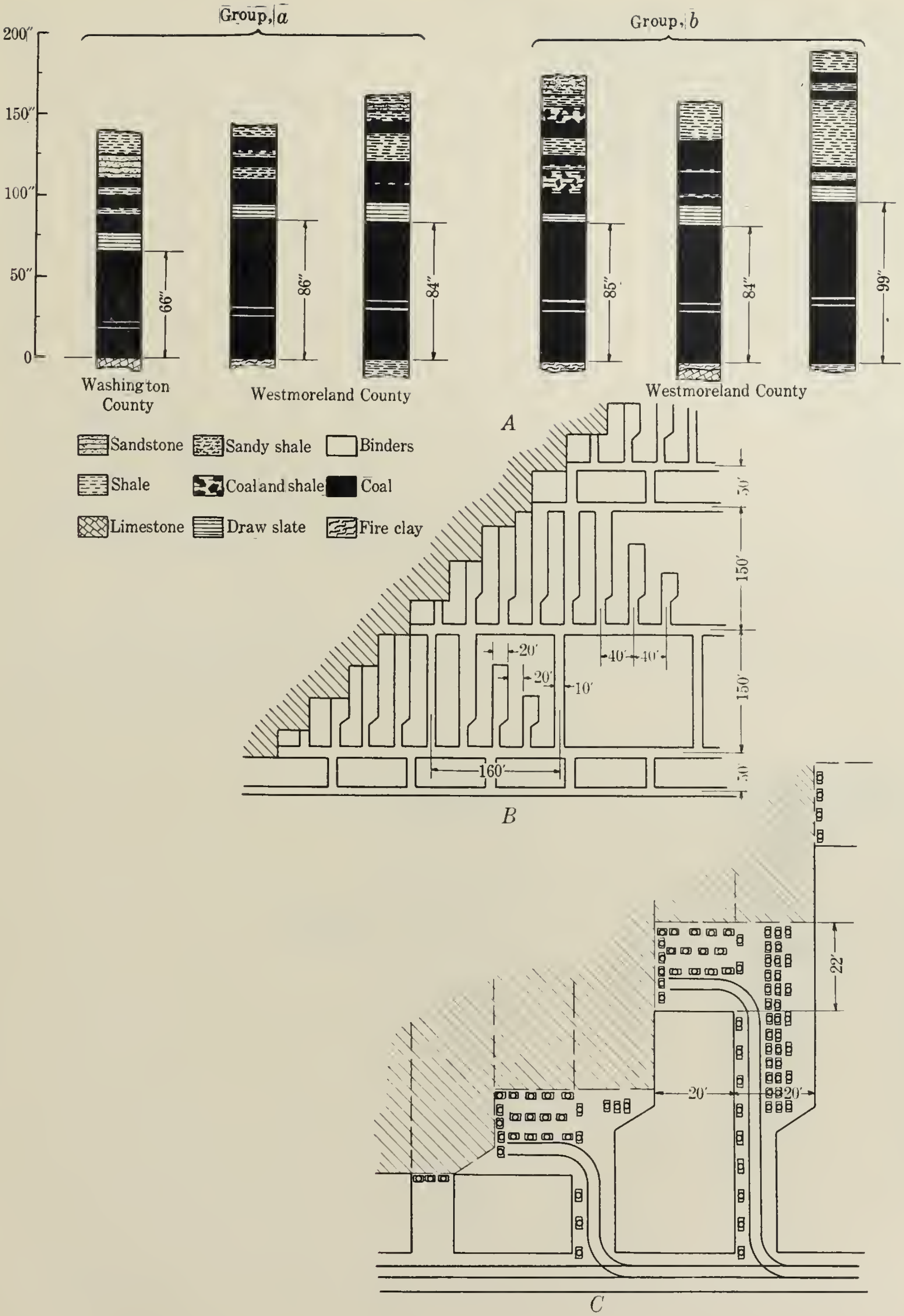

Figure 12.- Coal and roof sections, details of development, pillar extraction, and timbering for mines of group A, type E, in Washington and Westmoreland Counties, Pa.: A, Representative sections of the coal bed and immediate roof for mines in groups A and B of type E located principally in Westmoreland County, Pa.; B, development for pillar line using a subbutt heading, mine in Westmoreland County, Pa.; C, details of pillar extraction and timbering in pillar places for mine illustrated in B.

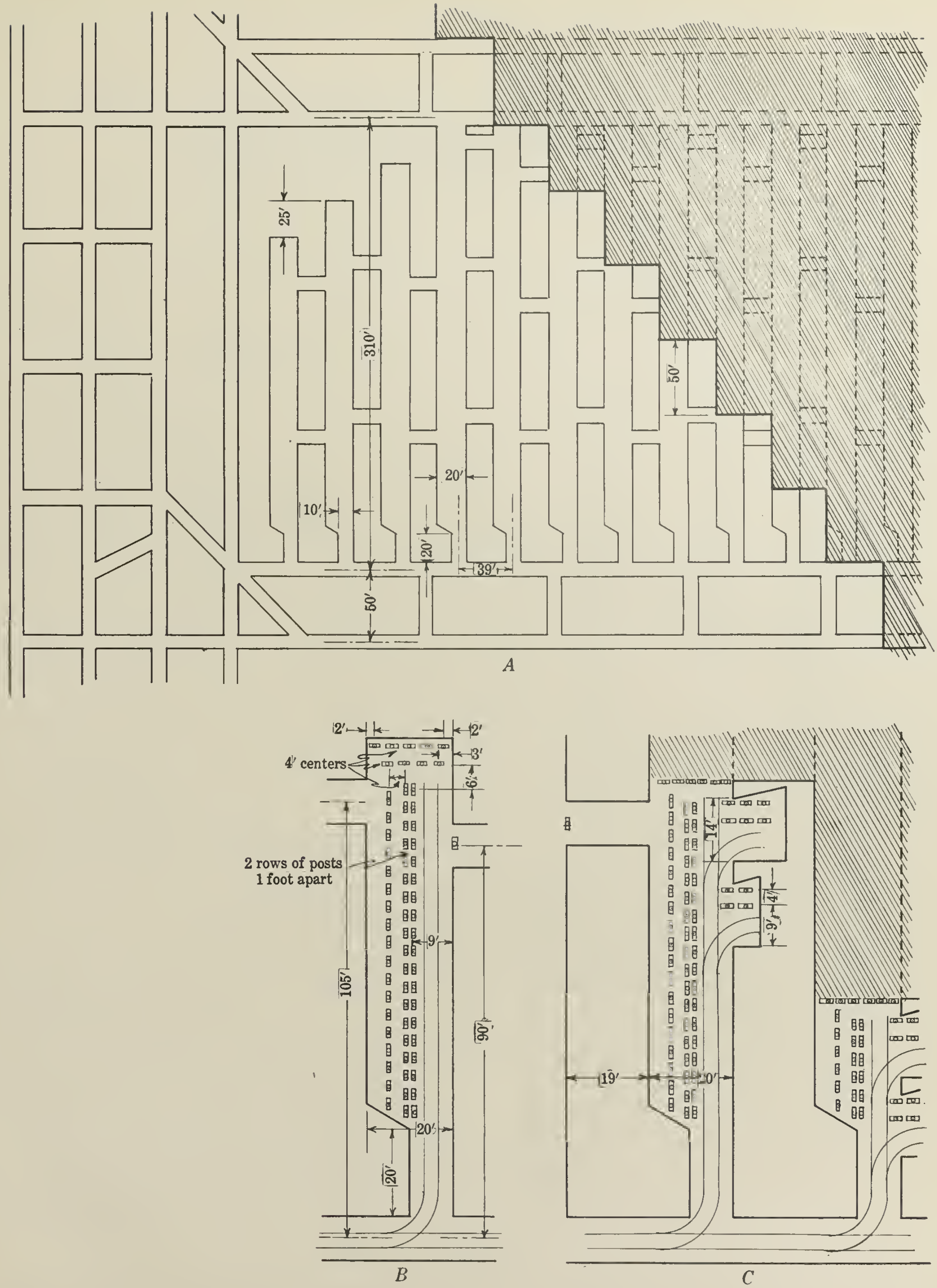

Figure 13.- Details of development, pillar extraction, and timbering for a mine in Westmoreland County, group A, type E: A, Details of development and pillar extraction; B, plan of posting roof and draw slate in rooms; C, plan showing sequence of operations in room-pillar recovering and timbering.

TABLE 5.- Developmental distances and operating results for a group of six mines using a full-retreat system with narrow room pillars. Group A of type E

Developmental distances	Representative mine (Mine 9)	Range for six mines		
		Max.	Avg.	Min.
Range of cover ... feet	[1]400-300-80	550	240	10
Width of headings ... do.	10	10	10	10
Number of headings on main entries ...	4	4	4	2
Number of headings on face entries ...	3	4	3	2
Heading centers on mains ... feet	60	60	52	40
Heading centers on faces ... do.	60	60	53	50
Heading centers on butts ... do.	50	60	52	50
Thickness of chain pillars on mains ... do.	50	50	42	30
Thickness of chain pillars on faces ... do.	50	50	43	40
Thickness of chain pillars on butts ... do.	40	50	42	40
Distance between face entries ... do.	1450	2400	1287	700
Center to center distances of butt entries ... do.	250	400	307	250
Barrier pillars on mains ... do.	200	200	156	75
Barrier pillars on faces ... do.	200	200	144	75
Width of rooms ... do.	18	21	19.5	18
Width of room pillars ... do.	15	29	18.5	13
Room centers ... do.	33	50	38	33
Length of rooms ... do.	200	320	260	200
Length of room neck[2] ... do.	15	30	18	13
Width of pillar pocket ... do.	12	21	14	10
Width of pillar fender ... do.	4	10	5.4	4
Angle of retreating rib lines with respect to butt entries ... degrees	45	52	45	24
Operating results				
Recovery ... percent	90		[3]88	
Coal produced per day per man underground ... tons	5.14		5.89	
Coal produced per post used ... do.	4.68		4.76	
Coal produced per pound of explosive used ... do.	No information			
Fatality rate per million tons of coal produced ...	.86		1.69	
Frequency rate for all injuries underground ...	27.74		128.73	
Severity rate for all injuries underground ...	.21		9.13	

[1]Three values represent maximum, average, and minimum values.
[2]Width of room neck same as for headings.
[3]For three mines only.

Group B of Type E

Group B comprises eight mines in Westmoreland County. This group differs from Group A principally in that rooms are driven 10 to 20 feet wide on 60- to 100-foot centers, thus leaving pillars 44 to 90 feet thick. There is no definite information on recovery for this group, but it probably ranges from 80 to 90 percent. Three of the mines produce 1,000 to 2,000 tons daily and five produce 2,000 to 3,000 tons daily.

The Pittsburgh coal bed varies in thickness from 72 to 99 inches, with an average of 84 inches for the eight mines. The draw slate averages 10 inches and the roof coal averages 11 inches in thickness. The main roof varies from a shale to a hard sandy shale. Between the roof coal and the main roof 30 to 70 inches of laminated coal and shale are found. Figure 12,A, shows representative coal and roof sections. In general, the coal is flat, although there are extensive areas in these mines where the dip is as much as 6 percent, and some variations are made in development on account of these grades. Rooms are driven on the face, but fairly often the main haulage roads and face entries are driven without respect to cleats to obtain better haulage grades. The average cover at the individual mines ranges from 250 to 350 feet, the maximum cover being 500 feet at one mine.

Figure 10,B, is representative of the general plan of development of seven of the mines of this group; figure 11,B, is representative of the eighth mine. Table 6 gives details of developmental distances and operating results. At seven of the mines, room pillars are recovered by driving butt pockets only off the rooms. This method is illustrated in figure 14,A, which shows the extraction of a butt pocket 14 feet wide with a 14-foot fender. After the pocket is completed the fender is mined by secondary pockets and the small secondary fender is mined by pick. At one of the mines where the pillars are developed in blocks 90 feet square, alternate butt and face pockets are driven off rooms and crosscuts, respectively Figure 14,B, shows six of the steps in the sequence of mining pillars by this method.

At two of the mines horsebacks in the roof are fairly prevalent and add to the roof hazards. They parallel the butt cleats, and therefore if rooms were driven on the face cleats they would extend the length of the room. Consequently, the development is planned so that rooms are driven on the butt cleats and thus cut across the horsebacks at right angles After a room is developed, the position of all horsebacks is known and the pillar pockets then can be driven to avoid them. This is an example of the application of good engineering judgment to the reduction or minimizing of natural hazards and also to the reduction of cost of mining. Another example of good engineering judgment in this group of mines is the method of development used in those parts of the mines where the dip of the coal bed is as much as 6 percent. Figure 14,C, shows the development for these conditions. By driving the rooms up the pitch rather than at right angles to the pitch, both drainage and cutting are facilitated, and the mining of the rooms and room pillars can be as rapid as with normal conditions, and a uniform rib line can be maintained.

At one of the mines, rooms were driven on 120-foot centers as the butt entries were developed, but when the rib line retreated, much of the roof in these rooms had fallen. The pillars were then split by new rooms on 60-foot centers and, in some cases, old rooms which had not caved too badly were cleaned. The fact that rooms were driven on 120-foot centers enabled the management to get out of a difficult situation, but this shows the advisability of not driving any rooms too far ahead of the retreating rib lines.

Figure 15,A, shows the method of extracting chain and barrier pillars. Figure 15,B and C, shows a representative plan of room timbering and timbering in a pillar pocket.

At some of these mines all working places are driven on sights. From 40 to 90 percent of the production is machine mined, and at three of the mines the coal is top cut.

Group C of Type E

Group C is composed of seven mines in Fayette, Greene, and Washington Counties; six are within a radius of 12 miles of Brownsville and one is near Washington, Pa. The type of development is similar to group B except that in the extraction of pillars alternate butt and face pockets are used more frequently. Recovery is high and averages 85 percent. These are all large mines, four producing 2,000 to 3,000 tons, one 3,900 tons, and the other 5,100 tons per day.

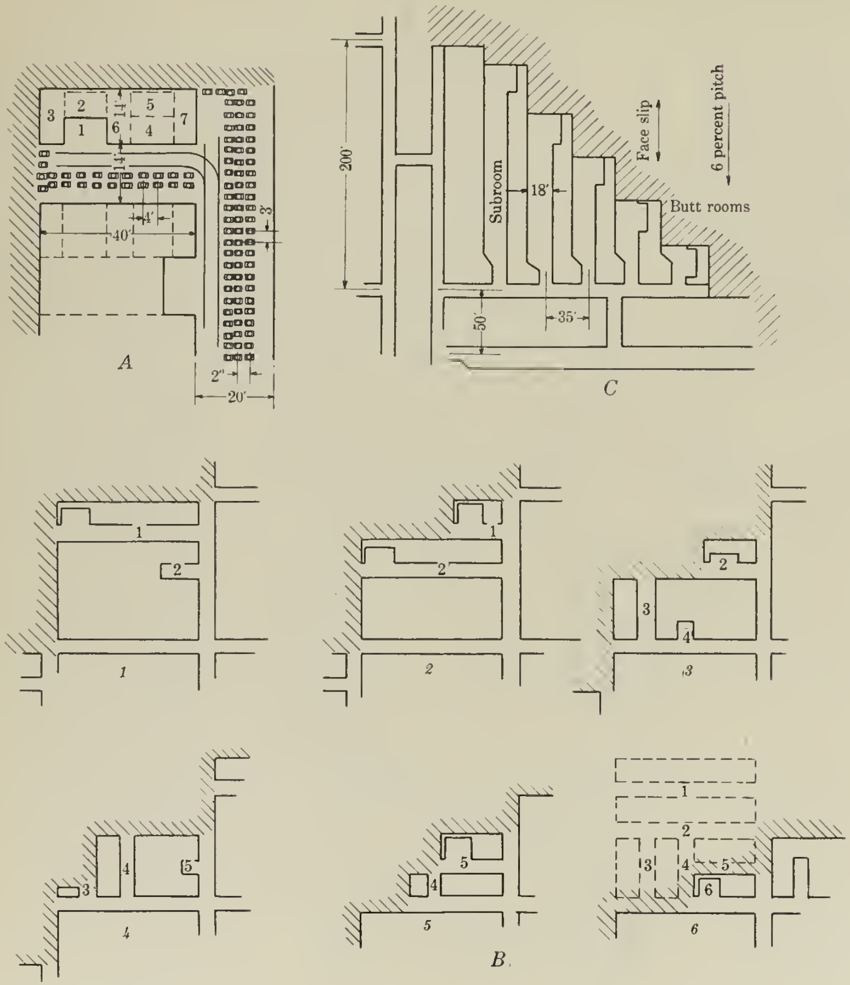

Figure 14.- Details of pillar extraction and timbering in pillar work for mines in Westmoreland County, group B of type E: A, Plan showing sequence of removing room pillars and position of timbers; B, six steps in sequence of operations in extracting a 90-foot block of coal; C, modification of regular system of mining by driving subrooms to avoid any difficulties from local pitches of the coal bed.

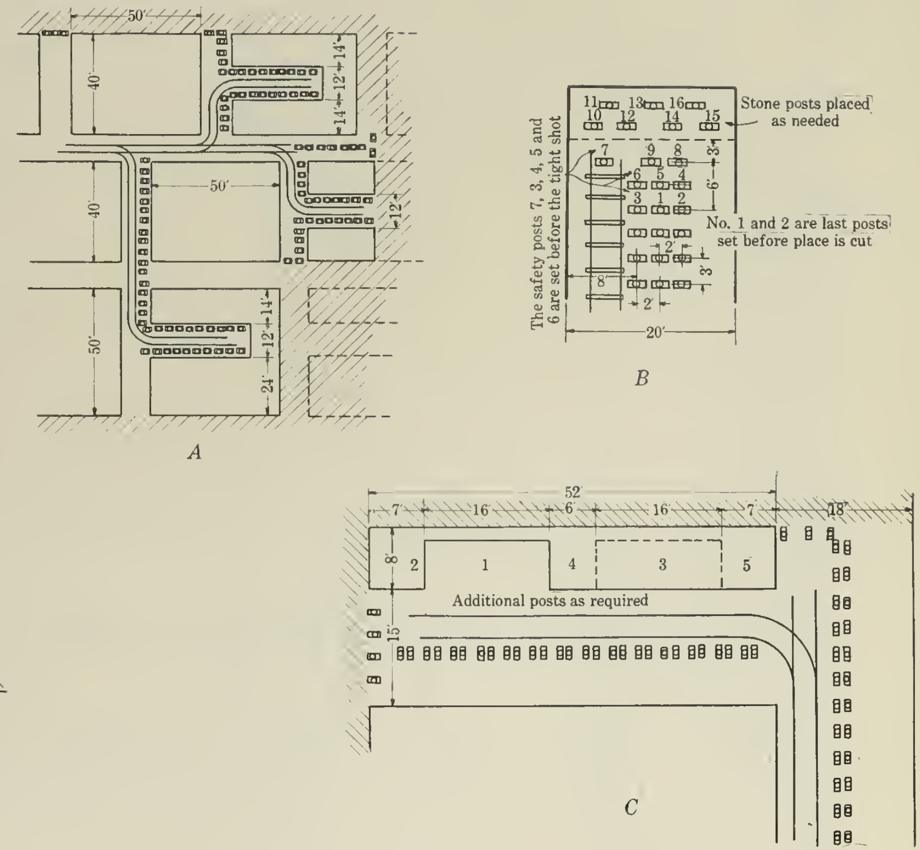

Figure 15.- Details of pillar extraction and timbering mines, group B of type E: A, Plan showing sequence of removing entry chain pillars and barrier and entry stumps between caved areas; B, sketch showing position of timber upon completion of loading the coal in a room (numbers indicate order of placing timber); C, plan of room-pillar process of removal, with location of timber and breakline posts.

TABLE 6.- Developmental distances and operating results for a group of eight mines in Westmoreland County using a full-retreat system with wide room pillars. Group B of type E

Developmental distances	Representa- tive mine (Mine 10)	Range for six mines		
		Maximum	Average	Minimum
Range of cover[1] ... feet	[1]300-250-200	500	294	10
Width of headings ... do.	10	10	10	10
Number of headings on mains ...	7	7	4	2
Number of headings on faces ...	2 and 3	6	3	2
Heading centers on mains and faces ... feet	50	100	56.5	40
Heading centers on butts ... do.	50 and 60	100	54	40
Thickness of chain pillars on mains and faces ... do.	40	90	46.5	30
Thickness of chain pillars on butts ... do.	40 and 50	90	44	30
Distance between face entries ... do.	1000	2000	1059	700
Center to center distances of butt entries ... do.	410	410	297	250
Barrier pillars on mains ... do.	50 to 90	200	120	50
Barrier pillars on faces ... do.	80	100	89	70
Width of rooms ... do.	20	20	17	10
Width of room pillars ... do.	40	90	54	40
Room centers ... do.	60	100	73	60
Length of rooms ... do.	340	340	250	160
Length of room necks[2] ... do.	21	40	20	15
Width of primary pillar pocket ... do.	14 to 16	18	14.5	10
Width of primary pillar fender ... do.	14	14	11.8	7
Width of secondary pillar pocket ... do.	12	24	18.4	12
Width of secondary pillar fender ... do.	5	8	5	3
Angle of retreating rib line with respect to butt entries ... degrees	45	60	45	17
Operating results				
Recovery ... percent	[3]No accurate information			
Coal produced per day per man underground ... tons	7.6	10.0	6.8	5.44
Coal produced per post used ... do.	9.6	9.6	6.5	2.8
Coal produced per pound of explosive used ... do.	11.3	13.8	9.4	5.5
Fatality rate per million tons of coal produced ...	4.87	4.87	1.97	...
Frequency rate for all injuries underground ...	252	252	101.9	48.2
Severity rate for all injuries underground ...	21.0	25.8	15.25	4.24

[1]Three values represent maximum, average, and minimum values.
[2]Room necks same width as entries.
[3]See text.

The Pittsburgh coal bed varies in thickness from 66 to 101 inches, being thickest near Uniontown and thinning out toward Washington. The draw slate varies in thickness from 5 to 36 inches and averages 13.8 inches. The mines east of the Monongahela river have no roof coal, but in its place have a black shale of variable thickness (9 to 60 inches); west of the river the roof coal is usually present and averages 12 inches in thickness. The main roof varies from a sandstone to a shale. Figure 16,A, gives representative sections of the

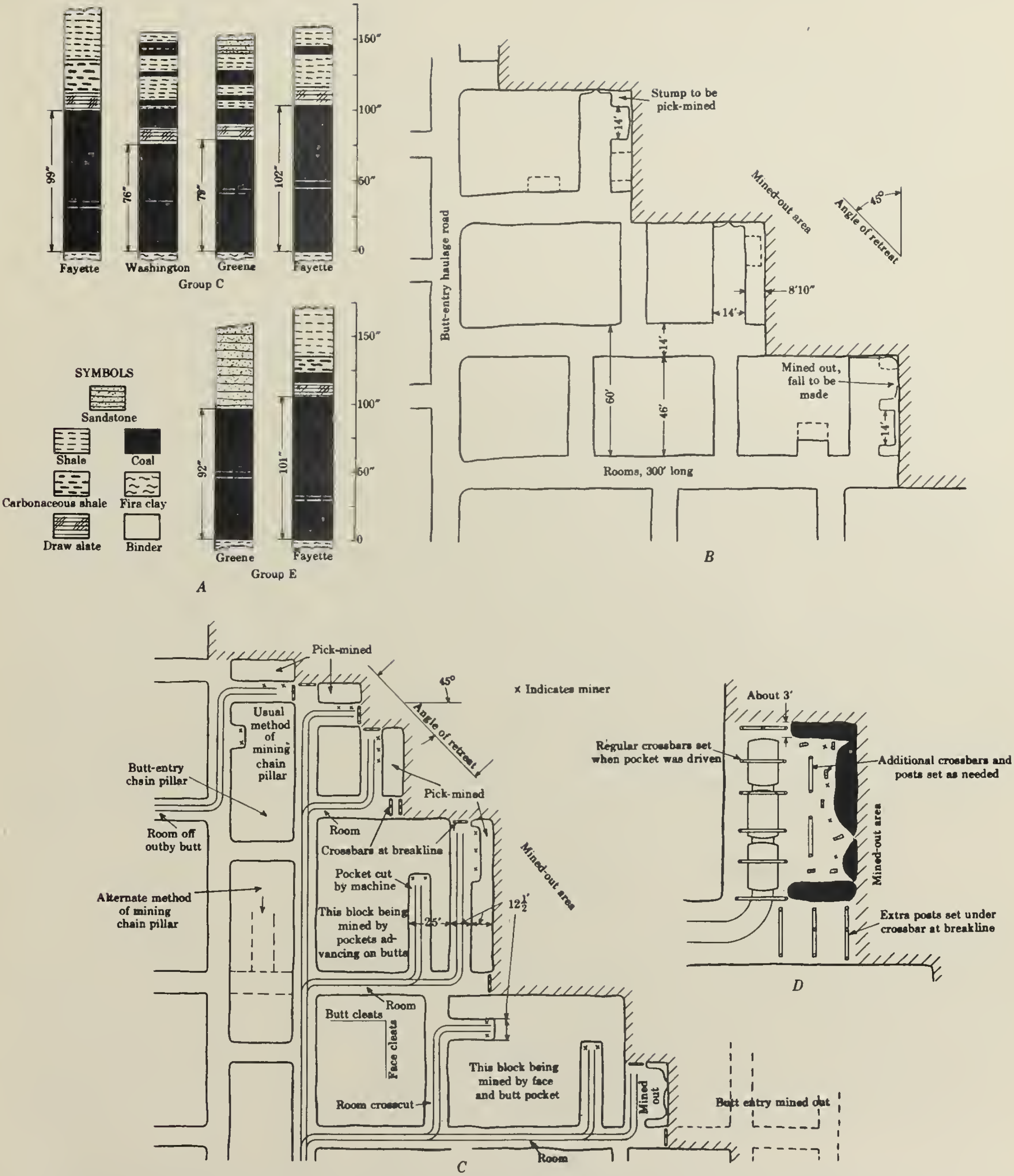

Figure 16.- Coal and roof sections and methods of pillar extraction for mines in Fayette, Greene, and Washington Counties, Pa., group C, type E: A, Representative sections of coal bed and immediate roof, groups C and E, type E mines; B, method of pillar extraction by butt pockets only for rooms on 60-foot centers; C, method of pillar extraction for rooms on 100-foot centers; D, details of timbering in pillar pockets (same mine as shown in 16 C).

coal bed and immediate roof in Fayette, Greene, and Washington Counties, in which counties are the mines of groups C and E, type E. The average cover varies from 300 to 625 feet with a maximum of 800 feet at one mine.

Figures 10,B, 11,A, and 11,B, show the general plan of development for this group of mines. Table 7 gives the details of development distances and operating results.

The general practice is to have the rib lines retreat at an angle of 45° with respect to face and butt entries. Figure 16,B, shows the details of pillar extraction for a mine with rooms on 60-foot centers. All pockets are driven on butt cleats off the rooms. Figure 16,C, shows the details of extraction for a mine with rooms on 100-foot centers. At this mine some pillars are recovered with butt pockets only and in some cases both butt and face pockets are used. Figure 16,D, show details of timbering in a pillar pocket at this same mine. Figure 17,A, shows the details of pillar extraction for a mine with rooms on 100-foot centers but with the pillaring line at an angle of 22-1/2° to the butt entries.

An important feature of type E is that barrier pillars, chain pillars, and room stumps can be recovered. Whenever possible, barrier pillars are mined in step with the pillaring line of the adjoining panel as shown in figure 17,B. Figure 17,C and D, shows the usual methods of recovery of chain pillars. While Figure 17,B, C, and D, shows the practice in mines of group C, the same general practice is followed at all type E mines.

At one of the mines butt entries are turned on 600-foot centers off the face entries. After leaving a barrier pillar of 200 to 250 feet, subfaces are driven with an intermediate butt entry turned off these subfaces on 300-foot centers. This method of development is shown in figure 18,A; it has the advantage of assisting ventilation and concentrating haulage, particularly as the rib line retreats along the barrier pillar.

At these mines 6 to 12 inches of top coal is left in place to help support the draw slate. Timbering in rooms and pillar pockets consists only of posts in some mines, of crossbars supported by posts in other mines and in some cases of a combination of crossbars and posts. Figure 18,B, C, and D, shows details of timbering for rooms and pillar pockets. Whenever the pillar pockets are driven the same width as the rooms, the method of timbering is usually the same in both. With the system of mining represented by type E, systematic and complete roof support in all working places is essential for safety and high recovery. Figure 19 shows further details of timbering in pillar work. From 59 to 90 percent of the coal is machine mined. At two of the mines the coal is top cut.

Group D of Type E

All of the mines studied in Monongalia, Marion, and Harrison Counties of the Fairmont district of northern West Virginia are included in group D. The methods of development are similar to groups B and C, but these mines are discussed separately, principally on the basis of geographical situation. Rooms are developed on 50- to 100-foot centers and the pillars are recovered principally by alternate butt and face pockets. Recovery is generally high and averages 83 percent. Fourteen mines were studied; three produced less than 1,000 tons, four between 1,000 and 2,000 tons, five between 2,000 and 3,300 tons, and two 6,000 tons per day, by working two shifts daily.

The Pittsburgh coal bed varies in thickness from 85 to 114 inches and averages 100 inches in the 14 mines studied. Based on the mines observed, the average thickness in each County is 93 inches in Monongalia County, 100 inches in Marion County, and 102 inches in Harrison County. The draw slate is variable; it averages 16 inches but varies from 2 to 48 inches and sometimes is entirely missing. The roof coal is also variable; it averages 8-1/2 inches but varies from 1 to 14 inches and sometimes is missing. The main roof varies from a thick shale to a sandstone. Figure 20,A, represents typical coal and roof sections in the Fairmont district. The average cover varies from 150 to 525 feet with a maximum cover of 900 feet at two mines.

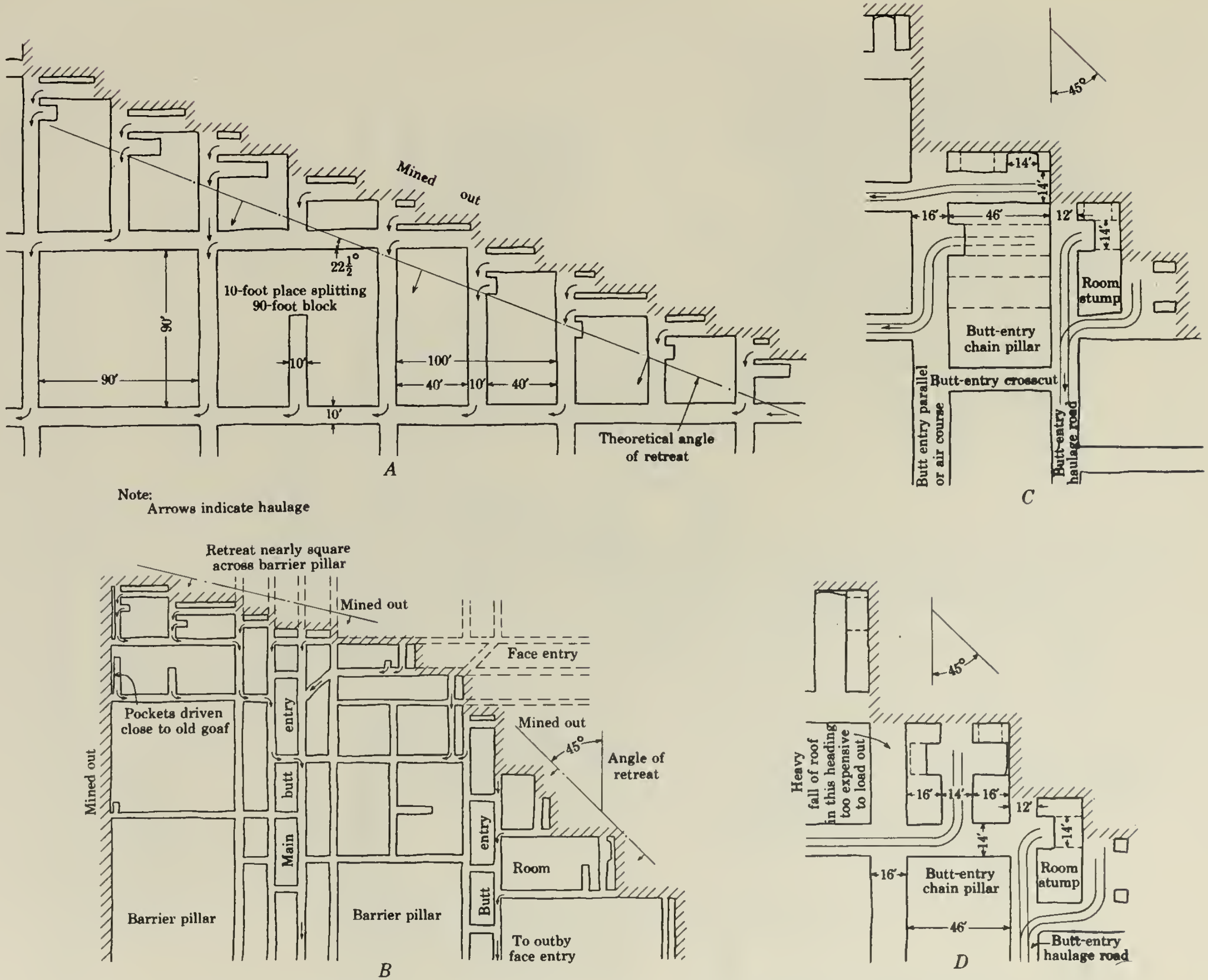

Figure 17.- Details of pillar extraction, group C, type E mines: A, Pillar recovery for rooms on 100-foot centers with pillaring line at angle of 22 1/2 °; B, method of mining barrier pillars in step with pillaring line in adjoining panel; C, general method for recovery of butt-entry chain pillars and room stumps; D, special method for recovery of chain pillars and room stumps.

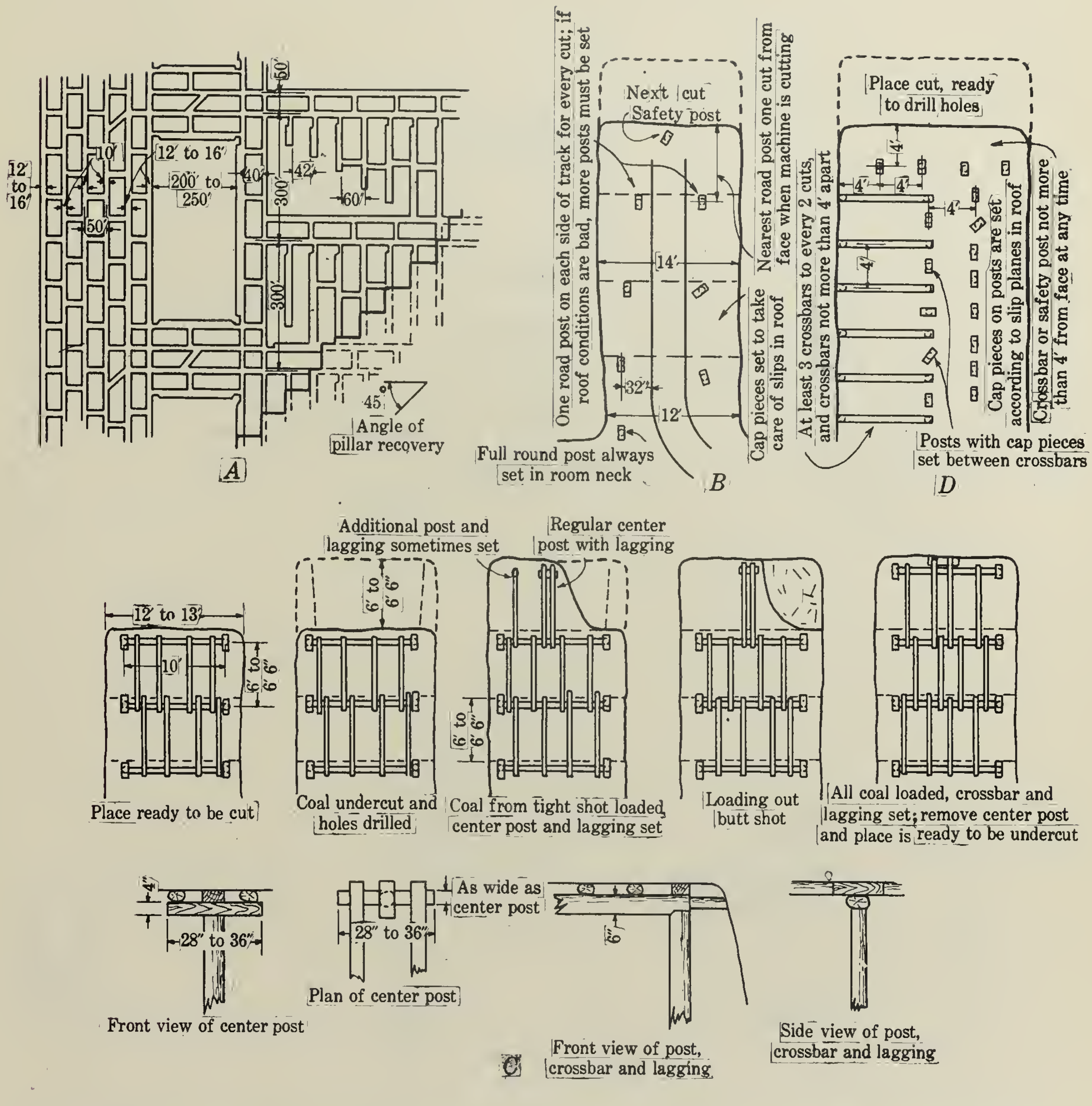

Figure 18.- Details of development and room timbering, group C, type E mines: A, Plan of development using subface and subbutt entries; B, room timbering using posts only; C, room timbering using crossbars supported by posts; D, room timbering using crossbars supported by posts and props.

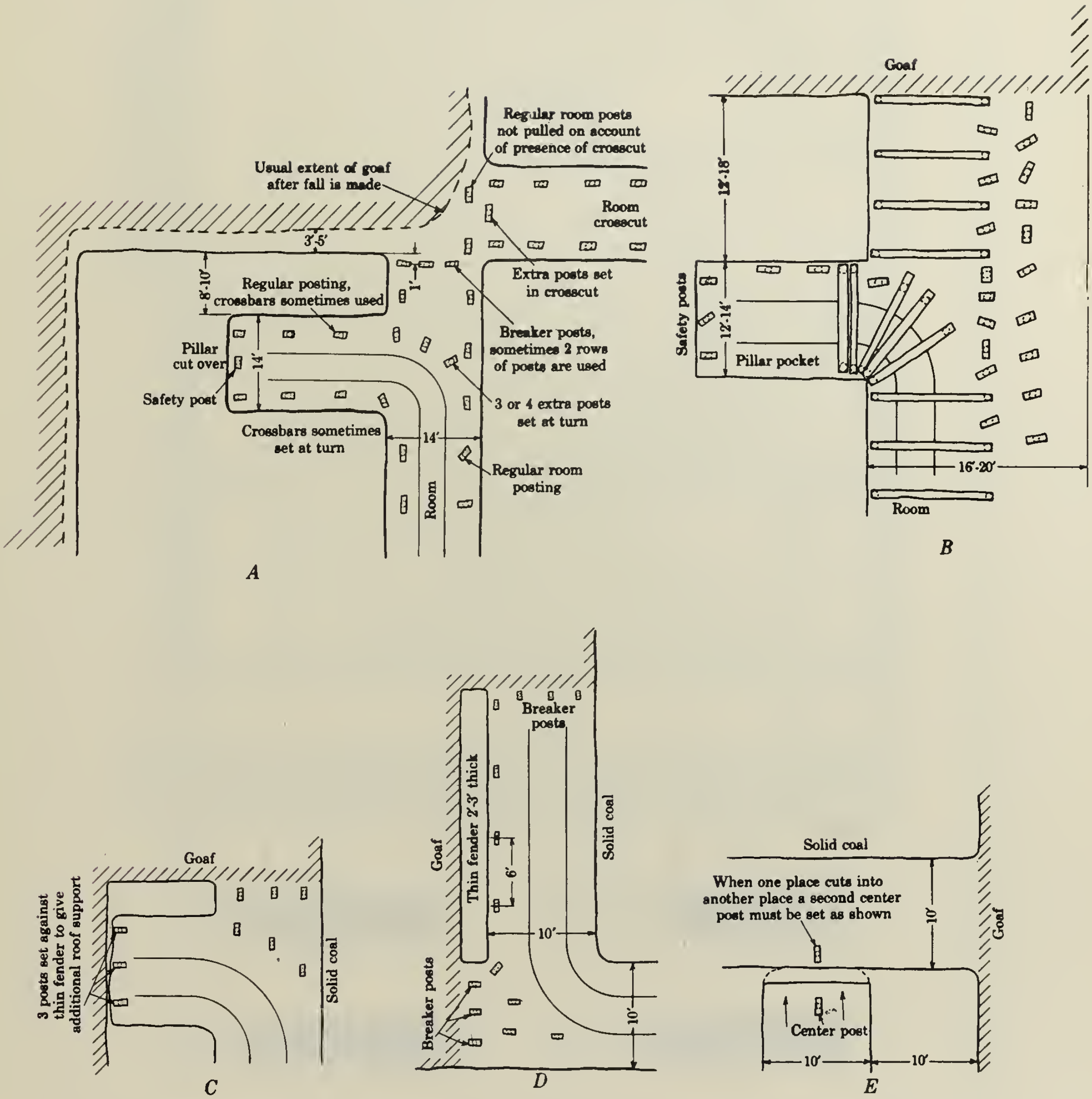

Figure 19.- Details of timbering in pillar work, group C, type E mines: A, Timbering in pillar places using posts only; B, timbering with crossbars and posts in pillar places; C, details of pillar timbering at goaf line; D, details of pillar timbering with thin fenders; E, details of pillar timbering when cutting into adjoining room.

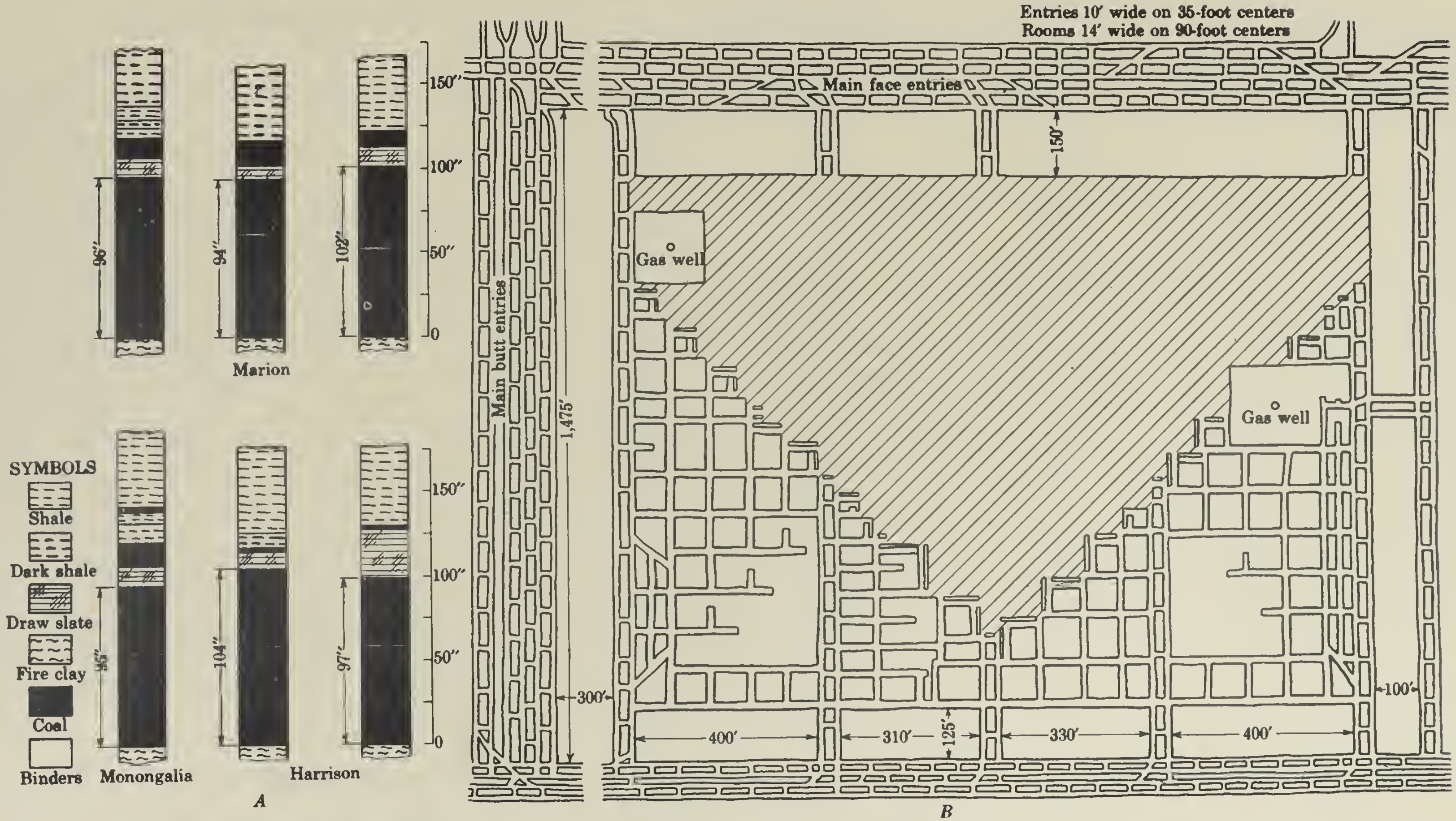

Figure 20.- Coal and roof sections in the Fairmont district of West Virginia and a special method of development, group D, type E mines; A, Representative sections of the coal bed and immediate roof in Harrison, Marion, and Monongalia Counties, W. Va.; B, special plan of development and pillar extraction to increase production from a panel.

TABLE 7.- Developmental distances and operating results for a group of seven mines in Fayette, Greene, and Washington Counties, Pa., using a full-retreat system with wide room pillars. Group C of type E mines

Developmental distances	Representative mine using 60-foot room centers, (Mine 11)	Representative mine using 100-foot room centers, (Mine 12)	Range for seven mines: Maximum	Range for seven mines: Average	Range for seven mines: Minimum
Range of cover ....feet	550-450-270	530-310-200	800	419	253
Width of heading ....do.	12	12	14	11.7	10
Number of headings on mains	4 and 5	4 and 5	7	5	3
Number of headings on faces	3 and 4	3 and 4	6	4	3
Heading centers on mains, faces and butts ....feet	60	50	100	60	50
Thickness of chain pillars on mains, faces and butts ....do.	48	38	90	48	37
Distance between face entries ....do.	1,400	1,100 to 1,600	2,400	1,418	1,100
Center to center distances of butts ....do.	360	300	360	337	300
Barrier pillars on mains ....do.	120 to 240	150 to 500	500	226	100
Barrier pillars on faces ....do.	140 and 200	170 and 200	450	195	100
Width of rooms ....do.	14	12.5	18	14.5	10
Width of room pillars ....do.	46	87.5	90	73	42
Room centers ....do.	60	100	100	88	60
Length of rooms ....do.	300	250	300	282	240
Length of room necks[2] ....do.	....	....	21	20	18
Width of primary pillar pocket ....do.	14	12.5	18	13.5	10
Width of primary pillar fender ....do.	8 to 10	12.5	18	9.8	3
Width of secondary pillar pocket ....do.	14	[3]15 to 25	18	(3)	10
Width of secondary pillar fender ....do.	5	3	5	(3)	3
Angle of retreating rib line with respect to butt entries ....degrees	45	45	45	45	22-1/2
Operating results					
Recovery ....percent	87.5	75		84.7	
Coal produced per day per man underground ....tons	5.7	5.2		6.2	
Coal produced per post used ....do.	6.68	3.72		5.2	
Coal produced per pound of explosive used ....do.	6.12	7.72		8.45	
Fatality rate per million tons of coal produced	1.41	0.54		2.79	
Frequency rate for all injuries underground	67.72	64.67		113.84	
Severity rate for all injuries underground	7.38	6.13		15.94	

[1]Three values represent maximum, average, and minimum values. [2]Width of room neck same as entries except for rooms 14 feet or less in width; room neck is same width as room. [3]Size of secondary pillar pocket and fender varies depending on roof conditions at the time the pillar is mined.

Figure 11,A, is representative of the general plan of development for mines of this group. Table 8 gives the details of developmental distances and operating results for two typical mines and the average for the 14 mines. Figure 20,B, shows a method for working two rib lines in one panel to increase the rate of production from the panel. At all of these mines the retreat of the pillaring lines is at an angle of 45° to the butt entries.

Figure 21,A, shows the usual method of pillar recovery and timbering in pillar pockets at those mines where the pillar blocks are mined by alternate butt and face pockets. Note that the fenders are 6 feet wide for both butt and face pockets. Figure 21,B, shows a variation in that fenders for face pockets are only 2 feet wide, whereas fenders for butt pockets are 12 feet wide. It is claimed that by taking into consideration the effect of cleats on the strength of the fender, better roof control along the pillaring lines is obtained. When pillar blocks are mined by butt pockets only, the methods used are similar to those described fcr groups A, B, and C of type E. Figure 21,C, shows a slight variation in that the 14-foot fender is slabbed by two machine cuts. Timbering in solid work and pillar places consists largely of posts only, but in some mines crossbars are also used. Top coal is usually left in place to help support the draw slate.

From 70 to 100 percent of the coal is machine mined, the average being 84 percent for all the mines. At three of the mines the coal is top cut.

Group E of Type E

The six mines in group E are developed in essentially the same way as those of groups B, C, and D; they differ, however, in the use of the open-end system of pillar extraction. The estimated recovery for these six mines is 87 percent. They are all in Pennsylvania, one in Greene County, four in Fayette County, and one in Westmoreland County. Three of these produced between 1,000 and 2,000 tons, one between 2,000 and 3,000 tons, one between 3,000 and 4,000 tons, and one produced 5,000 tons daily.

The coal bed varies in thickness from 78 to 104 inches and averages 93 inches. The draw slate varies from 8 to 18 inches with an average of 11-1/2 inches for five mines; at the sixth mine the immediate roof is a thick sandstone. Above the draw slate the usual roof coals and shales occur. The main roof at five of the mines is shale. Figure 16,A, shows some representative coal sections and immediate roof for these mines. The average cover varies from 250 to 740 feet with a maximum cover of 815 feet at one mine.

Figures 10,B, and 11,A, are representative of the general plan of development for mines of group E except that pillars are recovered by the open-end system. Table 9 gives development and operating results. In general, the pillaring lines retreat at an angle of 45°, but at one mine the retreat is at an angle of 34° with respect to the butt entries.

The general plan of development and timbering in solid places is similar to that discussed for group C of type E. The coal is developed by rooms and crosscuts into blocks varying in size from 80 to 125 feet square. Figure 22,A, shows the sequence used in mining blocks 65 by 100 feet in size. Figure 22,B, shows the sequence for blocks 88 feet square. Note that the block is first split by a 12-foot crosscut. The principle followed in all cases is to maintain a square block of coal, as the size of the block is reduced by alternate butt and face cuts. At some mines the pillar blocks are 68 feet square and these are mined by alternate butt and face cutovers. At one mine the pillar blocks are 112 feet square; on one side of the mine these blocks were mined by alternate butt and face cutovers, which made the first cutover 112 feet long, but on the other side of the mine the 112-foot blocks were first split by a crosscut, thus reducing the length of the first cutover to 48 feet. Chain pillars and barrier pillars are mined by open-end places in step with the main pillaring lines as described for group C of type E.

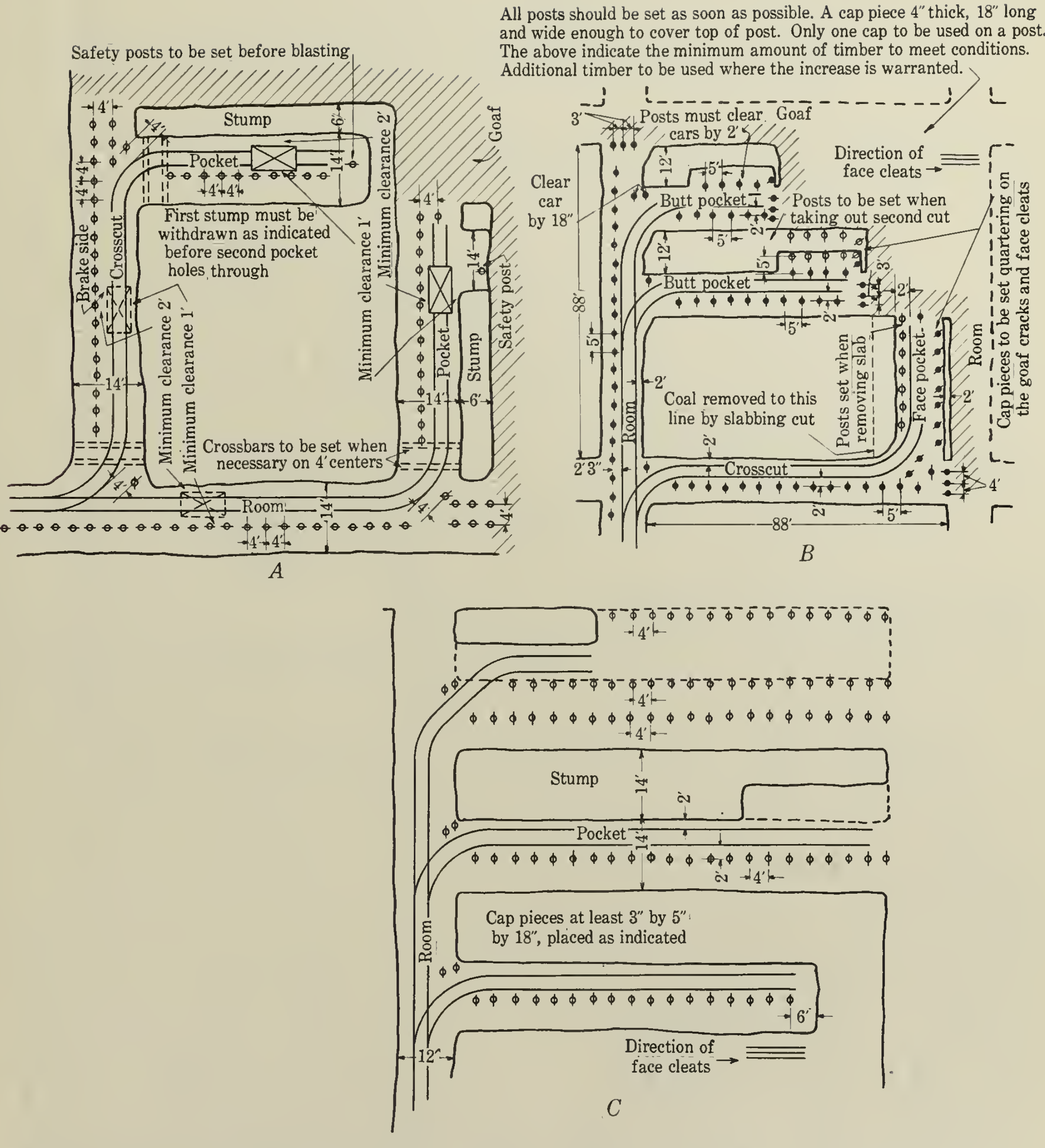

Figure 21.- Details of pillar extraction and timbering, group D, type E mines: A, Method of pillar extraction and timbering when mining 76-foot blocks of coal by alternate butt and face pockets; B, method of extraction of pillar blocks by alternate butt and face pockets, but with variation in the width of coal fender; C, method of extracting pillar blocks by butt pockets only, slabbing the fender by machine.

TABLE 8.- Developmental distances and operating results for a group of 14 mines in the Fairmont district of northern West Virginia. Group D of type E

Developmental distances	Representative mines		Range for seven mines with more than 200 feet of cover			Range for seven mines with less than 200 feet of cover		
	Moderate cover (Mine 13)	Light cover (Mine 14)	Maximum	Average	Minimum	Maximum	Average	Minimum
Range of cover[1] ... feet	750-450-250	400-150-20	900	450	20	600	175	20
Width of headings ... do.	12	12	12	11.8	10	12	11.5	10
Number of headings on mains	4 to 7	6	8	5	4	6	4	2
Number of headings on faces	3 or 4	4	6	4	3	6	4	2
Heading centers on mains and faces ... feet	100	35 and 62	100	59	35	62	45	35
Heading centers on butts ... do.	100	35 and 62	100	59	35	85	50	35
Thickness of chain pillars on mains and faces ... do.	88	23 and 50	88	50	23	50	33	23
Thickness of chain pillars on butts ... do.	88	23 and 50	88	50	23	73	36	23
Distance between face entries ... do.	1,700	1,200	2,250	1,756	1,200	1,800	1,400	1,000
Center to center distances of butts ... do.	400	370 and 470	420	373	330	500	411	250
Barrier pillars on mains ... do.	125 to 200	100 to 300	250	173	100	350	184	100
Barrier pillars on faces ... do.	125 to 200	150	200	161	100	150	130	100
Width of rooms ... do.	12	18	18	12.5	10	18	15	10
Width of room pillars ... do.	88	72	88	80	68	86	71	40
Room centers ... do.	100	90	100	93	80	90	86	50
Length of rooms ... do.	300	350	350	316	300	550	354	250
Length of room necks[2] ... do.	...	...	...	...	...	...	...	...
Width of primary pillar pocket ... do.	12	16	14	12	10	16	13	10
Width of primary pillar fender ... do.	12 or 2	7	18	8	2	8	7	6
Width of secondary pillar pocket ... do.	16 or 19	14	35	23	12	18	13	10
Width of secondary pillar fender ... do.	2	4	6	4	2	8	4	2
Angle of retreating rib line with respect to butt entries ... degrees	45	45	45	45	45	45	45	45
Operating results			Average for 14 mines					
Recovery ... percent	82.4	90	83.1					
Coal produced per day per man underground ... tons	6.88	5.28	6.76					
Coal produced per post used ... do.	10.5	13.23	14.17					
Coal produced per pound of explosive used ... do.	11.6	11.41	10.03					
Fatality rate per million tons of coal produced	1.93	1.67	1.86					
Frequency rate for all injuries underground	56.99	34.24	63.43					
Severity rate for all injuries underground	11.13	8.42	12.18					

[1]Three values represent maximum, average, and minimum values. [2]With narrow rooms the room necks are as wide as the room; in others, room necks are 15 to 20 feet long and as wide as headings.

TABLE 9.- Developmental distances and operating results for a group of six mines in the coking region of Pennsylvania using the open end system of pillar extraction. Group E, type E mines

Developmental distances	Representative mine, (Mine 15)	Range for six mines Maximum	Average	Minimum
Range of cover[1] ... feet	400-375-350	815	474	100
Width of headings ... do.	10 to 12	12	11.3	10
Number of headings on mains	4	7	5	4
Number of headings on faces	4	4	3	3
Heading centers on mains ... feet	30 to 50	85	64.5	50
Heading centers on faces ... do.	30 to 50	100	64	50
Heading centers on butts ... do.	40	125	72.5	40
Thickness of chain pillars on mains ... do.	20 to 40	73	53	38
Thickness of chain pillars on faces ... do.	20 to 40	88	52	38
Thickness of chain pillars on butts ... do.	30	113	56	28
Distance between face entries ... do.	2,000 to 5,100	3,100	1,940	1,000
Center to center distances of butts ... do.	340	500	356	250
Barrier pillars on mains ... do.	170 to 200	300	216	150
Barrier pillars on faces ... do.	170 to 200	300	200	150
Width of rooms ... do.	10 to 12	14	11.6	10
Width of room pillars ... do.	100	111	84	68
Room centers ... do.	112	125	96	80
Length of rooms ... do.	300	400	286	200
Width of open end pillar place ... do.	18	25	19.3	14
Maximum length of open end place ... do.	65	112	67	38
Operating results				
Recovery ... percent	85		86.7	
Coal produced per day per man underground ... tons	7.4		6.64	
Coal produced per post used[2] ... do.	10.2		6.77	
Coal produced per pound of explosive used ... do.	6.74		11.87	
Fatality rate per million tons of coal produced	0.64		2.74	
Frequency rate for all injuries underground	56.46		78.06	
Severity rate for all injuries underground	(3)		23.80	

[1]Three values represent maximum, average, and minimum values.
[2]The values represent timber for posts and crossbars only; no data are available for number of cribbing blocks used, but about 150 cribbing blocks are used per 1,000 tons of coal mined.
[3]Not available.

An excellent system of timbering is required if the open-end work is to be done safely and economically. Figure 22,C, shows the plan of timbering in an open-end place and figure 22,D, is a cross section of the open-end place giving further details of the timbering.

At one of these mines all coal is pick mined. At the other mines all coal is undercut, and at two of the mines it is center sheared in addition. No exact data are available on the percentage of machine-mined coal, but it is probably 90 to 95 percent, because all pillar coal is machine-cut except for small stumps at the end of pillar cutovers.

PRINCIPLES OF DEVELOPMENT AND PILLAR EXTRACTION

At some mines the details of development are planned as the coal bed and its condition becomes known, but it can be stated that a large percentage of the mines described in this report have been carefully planned by mining engineers who have considered all the factors which determine whether or not the mine shall be conducted efficiently. In deciding upon a system of development and the details to be followed, no one factor can be said to be most important; therefore, the following conditions should not be considered as of decreasing importance because of the order of their discussion.

Local customs and practices are often the basis for the choice of a particular system. This is justifiable to some extent because in any district the natural conditions for all mines are fairly similar. Therefore, the management of a new mine, knowing that nearby mines with similar conditions have used a particular system with success, follows the same system. Fairly often the opportunities for better recovery and more efficient operation are overlooked by not giving consideration to mining methods in other coal fields. Generally, the progressive mine management in any field is the one that breaks away from local customs and traditions and develops a system of mining based on all known factors, and after careful engineering planning gives the new method a fair trial under practical operating conditions.

Natural Conditions

Natural conditions must be considered. These natural conditions may be listed as depth of cover, character of immediate roof, dip of the coal bed, thickness of the coal bed, impurities in the coal bed and their location in the bed, the presence or absence of well-defined cleats in the coal, presence of gas, and probable amount of water to be handled. Frequently, mines are developed in direct opposition to the apparent dictates of these natural conditions. For instance, the dip of the coal bed apparently determines the position of the shaft or drift opening, but railroad facilities are not always available at the low point of the deposit and, further, the low point might be close to the property line and under deep cover, thus necessitating costly shaft sinking. Where facilities for river shipments are available, as along the Ohio and Monongahela Rivers, the economies of water transportation of the coal far offset the added cost of hauling against adverse grades in the mine. In some parts of Pennsylvania the coal is 7 to 9 feet thick and the top 10 to 12 inches high in sulphur. This coal is left in place to help support, with the addition of timber, the draw slate directly over it. This is a good example of leaving coal of low value in place to help increase the safety of the miner. The depth of cover is a natural factor which cannot be neglected, but it is not always the controlling factor. With increase in cover it is the general practice to increase the size of barrier and chain pillars and of room stumps and room pillars, but the converse is not true; that is, a system favorable for 400 to 500 feet of cover may be used without change for a cover of 150 to 300 feet, although it would appear that modifications would be possible for greater economy in mining. It is probable that other factors were deemed to be of greater importance and no changes were made.

The importance of cleats in the Pittsburgh coal is shown by the fact that the development of all these mines is planned so that the rooms will advance on the face cleats because the mining is easier; depending on the system used, from 65 to 100 percent of the coal is mined from places advancing on the face cleats.

Aim of Choice of Method

The choice of a method and its details should aim at (1) high recovery, (2) roof control, and (3) safety; and should provide for (4) economy by concentration, (5) proper venti-

lation, (6) efficient haulage, and (7) economical drainage and pumping of underground water. Any system that has been determined upon by careful engineering planning should be followed faithfully, but it should be flexible enough so that it can be altered in detail to suit the conditions as they change in the actual progress of the mining.

When the system of development has been determined upon, the next important consideration is whether or not pillars are to be recovered, and, if so, how shall they be mined. The value of the coal in place if the coal is owned by the mining company or the extraction in tons per acre required of the operator by the owner of the coal usually determines whether or not pillars are to be extracted. The necessity for protection to the owners of surface rights may also modify the problem; however, this is a legal matter and before the question of pillar extraction can be decided, the legal aspects of the relationships between the owners of the surface and the owners of the coal must be ascertained. It is generally believed that a greater percentage of lump coal results from solid work as compared to pillar work, but the strength of the coal and the method of pillar extraction also affect the percentage of lump production, as well as the method of drilling, cutting, and blasting. It cannot be said that better roof control results when pillars are not mined than when pillar recovery is attempted. Roof control depends upon the system of mining, but unless a high degree of skill in underground management is exercised the system of mining will not prevent the roof from getting out of control. Squeezes may develop as easily in mines where pillars are not extracted. It is essential, however, to use the system best adapted to the conditions. Neither can it be said that more timber is required when pillars are extracted, because the amount of timber required does not depend so much on the methods used as on the speed with which places are driven and abandoned - that is, on how much retimbering is necessary before all the coal is mined. In pillar work timbers are recovered to help roof breaks and, to a large extent, some of this timber can be used again. Recovery of timber in moderate to heavy cover is essential for roof control.

Developmental Costs

A factor of great importance insofar as costs are concerned is the amount of money that must be tied up in development work before normal production is reached. When this normal production is attained, the amount of development can be proportioned according to probable future demands. Few mining companies prepare any estimates of future demands. Such estimates, if considered only as estimates, should help guide the mine management in deciding upon the amount of development required. When demand slows up, many operators find their mines overdeveloped, which is costly; when demand picks up again, the lack of development results in a lack of ability to produce the desired amount of coal. These failures are to be attributed to lack of cooperation between the higher operating officials and the sales departments in not trying to form an estimate of the probable future demands based on the available knowledge of conditions and the past experience of these men in the coal industry.

For quick production with a minimum of development, the methods used in eastern Ohio and the Panhandle districts of Pennsylvania and West Virginia are ideal. However, if pillars are to be withdrawn, these methods are suitable only under shallow cover, and if pillars are not to be drawn, they are suitable under moderate to moderately heavy cover. From 30 to 40 percent of the coal is left in the ground to support the overburden however, and actually more development is required for a given production than is generally realized because of the large percentage of coal left in pillars.

The half-advance and half-retreat system provides for quick production, for the maintenance of a uniform daily tonnage, and for a minimum of development. The full-retreat method with short rib lines has the same qualities except that slightly more development is required

on account of the butt entries being fully developed before room-and-pillar work begins. Full retreat with long fracture lines requires the greatest amount of development before room-and-pillar production begins but, by maintaining an even balance between room-and-pillar work and new development, the latter need not be excessive but will be greater than with the other systems. The large number of mines using this method indicates that it has many advantages over all the other methods. These advantages, in properly managed mines, are (1) excellent roof control and (2) concentration of work. The latter includes concentration of haulage, track work, ventilation, and supervision; all these factors tend toward a lower production cost.

Size of Pillars and Stumps

In any system of development, the size of barrier pillars, chain pillars, and room stumps must be considered, and these are controlled to a large extent by the amount of cover. With the full-advance system, whether or not room pillars are extracted, the room stumps and chain pillars form a protection to the butt entry while the rooms are being worked. With a full-retreat system, these chain pillars and room stumps are mined in step with the room pillars, and in this case the chain pillars do not have to support the roof after coal has been mined out on each side of the butt entry. Some mine managers consider it good practice to drive butt entries on the same centers as rooms even though the rooms are on 100-foot centers. The chain pillars are thus developed in blocks of the same size as the room pillars and these pillar blocks are then extracted in the same order as room pillar blocks. Whether main haulage and face entry headings should be driven on 100-foot centers is questionable, but some mines follow this practice and find that when the time comes for ultimate recovery of these main entry chain pillars, the blocks are in good shape and the pillar recovery is as efficient as the recovery of room-pillar blocks.

The width of barrier pillars on main haulage roads and face entries is determined by the amount of cover, but where foresight has been used these barriers are made far wider than the amount of cover requires. In any case, the barrier pillars should be made 250 feet wide, or equal to the usual room length, to justify the expense of laying switches and track for the recovery of coal in a room 250 feet long as compared to a room only 50 or 100 feet long, as would be the case with thin barrier pillars. Some mines leave barriers as much as 500 feet wide, and some leave an entire panel unmined along the main haulage roads. This is an ultimate economy because, as the mine is worked out, large blocks of coal are available for retreat to the mine opening. Instead of being confronted with long strips of coal only 500 to 600 feet wide and cut up by four or more headings, the management has blocks of coal 1,000 feet or more in width. A long breakline can be established and a uniform system of pillar extraction can be followed as in a panel. Furthermore, a large daily production can be maintained almost to the day the mine is finally abandoned, which is economical when compared to the gradual falling off in production when only thin strips of barrier pillars remain to be mined. Whenever it is possible, the progressive mining companies mine the barrier pillars in step with the fraoture line of the last panel which these barrier pillars are protecting.

The size of the panel varies even in individual mines. In many mines this variation is due to the location of property lines and the outcrop of the coal bed. In some gassy mines the panel is not made large, so that by the time the development work is completed much of the gas has been drained off before pillar work begins. For any panel, the length of the fracture line will vary depending upon the stage of completion. Thus, at the start and at the finish of the fracture line the production from the rib line will be less than normal and provision must be made to take care of this variation in production by the development of other panels. The ideal arrangement is for a new rib line to begin about the time that an old rib line is working into the corner of its panel and its production gradually diminishing.

The essentials for good pillar work and roof control along pillaring lines are simple and consist of two things: (1) A minimum amount of development of entries and rooms ahead of the pillar line and (2) development of rooms as needed to reach the pillaring line at the proper time, and then immediate extraction of the pillar. These essentials imply a carefully planned system of development, pillar extraction, and timbering in both rooms and pillar work. It has been found that places advancing toward the gob have as good roof control as those advancing toward solid coal, and have no more, if not less, injuries from falls of roof. Opinion varies on the proper length of a rib line for its efficient supervision by the mine officials. Rib lines varying from less than 1,000 up to 4,000 feet in length have been observed. It is generally conceded that a rib line should not be any longer than can be properly supervised by one assistant mine foreman assisted by the fire bosses, rib bosses, and shot firers under his charge. About 1,500 feet would be the maximum length to suit this rule under the conditions encountered in mining the Pittsburgh coal bed.

COMPARATIVE EFFICIENCIES OF VARIOUS METHODS

In describing each type of development one or more mines were cited as representative of the system, although actually these individual mines are the best and most efficient operations of their type. These mines, all above the average in their respective fields, will be used in comparing the five types. The importance of strong financial backing necessary for efficient mine operation is shown by the fact that of these 12 mines seven are owned and controlled by strong companies in allied industries such as steel, power, and transportation, and the other five are owned by coal operating and sales companies generally considered as being financially strong.

Table 10 gives some values which offer a means of comparing the efficiencies of the various systems of mining. The values given mean little unless the conditions controlling them are understood. The factors affecting each item will be briefly discussed. In the first place, it should be evident that the methods of development and pillar extraction have little to do with the values as given in table 10. A much more important factor is the efficiency of all the details of underground operations such assupervision, haulage, cutting, blasting, timbering, and discipline.

Discussion of Operating Efficiencies

Labor Classification

Mine superintendents, mine foremen, assistant foremen, and fire bosses are all included under the classification of officials. Shot firers are not included in this class but are placed under "daymen", because although shot firers have certain supervisory duties, not all the mines employ shot firers. A high tonnage per day per official may mean that the mine workings are concentrated and that a few officials can cover the mine adequately, but on the other hand it may mean that the supervision is inadequate. In general, a low tonnage per official is characteristic of the more efficient mine, because the cost of additional supervision is more than returned in other economies which result from close management of underground operations.

Dispatchers, haulage bosses, motormen, trip riders, drivers, trap-door attendants, and bottom men are classed as haulagemen. Efficiency is therefore affected by the method of entrance to the mine, whether by shaft or by drift. Where gathering is by animals, the efficiencies will be lower and also the length of haul from the active workings to the mine opening will have its influence. However, since the majority of haulagemen are engaged in gathering and secondary haulage, the tons per day per haulageman is some indication of the degree of concentration of the mine workings.

TABLE 10.- Comparative operating efficiencies of representative mines using various types of development and pillar extraction

Item	Full advance: Without pillar recovery		Full advance: With pillar recovery	Full retreat with short pillar lines	Half advance and half retreat	Full retreat with long pillaring lines: Room centers less than 50 feet	Room centers more than 50 feet	Pillar blocks in Connellsville region		Pillar blocks in Fairmont region		Open-end pillar extraction
Type[1]	A		B	C	D	E-A	E-B	E-C	E-C	E-D	E-D	E-E
Mine number	1	2	3	5	7	9	10	11	12	13	14	15
Coal produced per day, tons:												
Per mine official	401	394	259	231	187	190	309	216	431	208	(2)	(2)
Per haulageman	63	77	65	100	57	59	95	46	108	78	(2)	(2)
Per machineman	71	112	60	196	99	72	412	169	185	130	(2)	(2)
Per miner	7.3	10.1	5.6	10.7	6.5	6.8	11.1	10.7	10.9	10.3	11.6	(2)
Per dayman	55	102	92	43	47	82	57	20	31	32	(2)	(2)
Per man underground	5.3	7.5	4.4	7.4	4.8	5.1	8.1	5.7	7.1	6.5	5.8	7.4
Active areas of exposure, square feet:												
Per ton of daily production	500	700	700	500	400	300	800	(2)	300	200	3,800	300
Per man underground	3,400	4,900	3,300	3,700	1,800	1,700	5,100	(2)	2,000	2,300	25,800	2,800
Production, tons:												
Per pound of explosive used	(2)	(2)	6.8	(2)	11.3	(2)	(2)	6.1	7.7	(2)	11.4	(2)
Per post used	[3]4.7	4.0	4.8	5.2	4.8	4.7	9.6	6.7	3.7	10.5	13.2	[3]3.0
Recovery of coal, percent	62.3	[3]65	[3]80	85	86	(2)	(2)	87	75	(2)	(2)	(2)
Accident rates:												
Fatalities per million tons	0.91	3.60	2.38	1.86	4.15	0	2.94	1.41	0.54	1.93	1.23	0.64
Frequency rate	179.7	26.0	123.0	60.4	196.6	27.7	252.2	67.7	64.7	57.0	34.2	56.5
Severity rate	9.2	25.9	13.6	10.8	21.4	0.2	21.0	7.4	6.1	11.1	8.4	(2)
Cost of compensation, cents per ton	3.8	3.0	4.4	1.1	3.6	(2)	(2)	1.3	1.1	2.4	2.4	(2)
Accident rates, average for each type:												
Fatalities per million tons	4.1		0.6	1.8	4.0	1.7	2.0	2.8		1.9		[4]2.7 [5]2.1
Frequency rate	126.0		157.3	80.8	142.0	128.7	101.9	113.8		63.4		[4]78.1 [5]62.0
Severity rate	21.9		9.2	10.4	18.8	9.1	15.3	15.9		12.2		[4]23.8 [6]14.5

1. As classified in this report. 2. Information not available. 3. Estimated. 4. For six mines. 5. For five mines. 6. For four mines.

Machinemen and their assistants are included under the classification of machinemen. Here again the tons per man is a measure of degree of concentration of work, because the shorter the moves from face to face the greater the tonnage of coal cut per shift. But many factors influence these values, such as the height of coal, whether track-mounted or shortwall machines are used, the amount of pick coal mined, whether the coal is bottom cut, top cut, or sheared, whether the machinemen also drill the shot holes, whether the coal is cut on the night shift, and whether the clean-up system used allows the loaders to load out a full cut daily or permits the miners to load coal at their own convenience, thus slowing up the tonnage per mining machine.

The tonnage per day per miner is a measure of the efficiency of that class of labor and also the skill of management in proper organization of the mine to serve the face worker. Factors affecting these values are as follows;- character of car supply, amount of slate to be handled, whether miners or machinemen drill the shot holes, the amount of timbering done by the miners, and the percentage of pick mined coal. At some mines the duties of the miners consist only of loading coal and setting the necessary timbers to protect themselves at the face. At other mines the loaders have to load coal, set timber, handle draw slate, drill holes, and lay track. Obviously, with such variation in the duties of a loader, tonnage per shift will vary, regardless of the methods of development and pillar extraction.

Under the classification of daymen are included timbermen, trackmen, pipemen and pumpmen, bratticemen, masons, and laborers. Here again tonnages per shift should be high if the work is concentrated, but a high tonnage may indicate that the proper maintenance of the mine is being neglected to reduce the immediate costs and, conversely, a low tonnage per shift may mean that the daymen are not properly supervised or that not enough men are being employed to maintain the mine in first-class condition. Natural conditions, such as character of roof and amount of water to be handled, particularly in working places, affects these results, although in respect to water, the efficient mines alter the regular method of development so that there will be a natural gravity drainage away from all working places.

The tonnage per day per man underground is a true measure of the efficiency of operation, including development, pillar extraction, haulage, supervision and safety. The values given in table 10 are based on single calendar years which were chosen to represent, as nearly as possible from the data available, years of about equal business activity for all the mines. Efficiencies may vary from year to year for individual mines; this was particularly true in 1929 and 1932. During the latter year much effort was made to distribute the small amount of work among as many men as possible, and under such circumstances there was a tendency toward reduction in tons per man underground. The general practice is to work one man in each solid place and two men in pillar places. This is varied according to the necessary speed of development or pillar extraction required to maintain uniform rib lines. As many as four to eight men may be worked on a pillar stump if its removal is urgently necessary.

Areas of Exposure

The areas of exposure listed in table 10 indicate the amount of exposed roof area to be maintained on the basis of the number of men underground and the average daily production. These values indicate that, in general, the mines extracting pillars have less unit area of exposure, but to a large extent this is a result of more efficient underground management. Areas of exposure are a poor indication of the efficiency of a method of development. A mine producing a maximum possible tonnage will have a minimum area of exposure, but if the production is curtailed, the unit areas will increase, and then it would appear that the system is not adaptable. As a mine grows older, the length of haulage roads increases, which adds considerably to the area of exposure. Efficient underground management can reduce areas of

exposure, regardless of the method of development. In eastern Ohio rooms are 24 feet wide, and with coal 5 feet high and a 6-foot undercut, each room should produce about 29 tons per shift. Therefore, when only one man works in each place with an average daily production of 8.7 tons, at least 230 places are necessary for the mine to produce 2,000 tons daily. In mines extracting pillars the daily production per loader ranges from 5.6 to 11.6 tons, and the average for nine of these mines, as given in table 10, is 9.4 tons. In these mines, assuming a place 12 feet wide, coal 6 feet high, and a 6-foot cut, each place should produce about 18 tons per shift. In normal times it is the practice at these mines to work two men in each place and load out a cut of coal at each face every day the mine works. Therefore, to produce 2,000 tons per day, only about 111 places are required to mine this production. These comparative values indicate that the kind of underground management is equal in importance to the methods of development, if the areas of exposure are to be kept at a minimum.

Explosives and Timber

The tons of coal produced per pound of explosive used may depend upon the amount of pillar work, but greater variation is caused by the kind of explosive used, the strength of the coal, whether or not the coal is undercut and sheared, the amount of pick work, and the demand for lump coal. Permissible explosives are generally used in these mines whether gassy or not, but in some mines black powder in pellet form is used.

The amount of coal produced per post used is higher in general when pillars are extracted than when pillars are not extracted. The character of roof is a factor which largely determines the amount of timber needed, but if two mines having similar roof conditions are compared, the amount of timber used will not depend so much on the method of development and pillar extraction as on the speed with which rooms are driven and pillars extracted. When working places remain idle, retimbering becomes necessary and thus the use of timber is greatly increased.

Coal Extracted

The data given on the percentage of coal recovered is not comparable in all cases. Some of the information is taken from estimates made by company engineers based on surveys, and some is merely the guess of mine officials. In table 10 the percentages of recovery as given for mines 1, 3, 5, 11, and 12 are engineering estimates. For the methods used in Ohio and the Panhandle district of West Virginia the recovery will range from 55 to 70 percent, and for mines recovering pillars from 75 to 95 percent. An important factor in high recovery is steady mine operation. Much coal is lost in pillar stumps when mines operate only one or two days a week.

Accidents

The data given in table 10 with respect to underground injuries indicate that the recovery of coal in pillars can be done safely. The problem of safe mining does not depend so much upon whether or not pillars are extracted but rather on the efforts of the management toward the enforcement of safe practices by all underground workers. In table 10 the accident rates are given for the 12 selected mines and also the average rates for each type. The rates for group E of type E are high because of the great number of injuries and fatalities at one of the six mines. It should be noted that mines recovering pillars have a lower fatality rate than those not recovering pillars, and in general the same holds true for the frequency and severity of all underground injuries.

Types C and E have these low rates because the method of development and pillar extractions requires close supervision, and it naturally follows that the safety of the worker at the face is also closely supervised. As laxity in management of development and pillar extraction cannot be tolerated with these types, it therefore follows that laxity in other details of underground mining, particularly with respect to safety, is not tolerated. A method of development requiring close supervision apparently induces the close supervision necessary to reduce injuries.

CONCLUSIONS

From the foregoing discussion it is evident that for an efficient mine the first essential consideration is the selection of a system of development and pillar extraction suitable to the natural conditions encountered; and the second essential consideration, of equal if not greater importance, is the efficient management of all the details of underground operations. Of the five types of development discussed in this paper, the degree of management necessary to control development increases from type A to type E. It is very probable that mines of type E are more efficient than those of other types for the simple reason that the method of mining must be closely supervised, and it is natural to expect that this keener supervision of development will also result in keener supervision of all the other details of mining. With type E it is essential that all mining operations proceed on a uniform schedule to maintain uniform pillaring lines. In types A and B uniform mining is not so essential; it can be neglected to some extent, and with it other details of mining are likely to be neglected by the management.

This discussion has dealt with engineering problems relating to the development of plans of mining which have different degrees of efficiency in operation and the recovery of coal, has brought to the attention the loss of valuable areas of coal that might be recovered by the adoption of methods which have proved successful in other fields having similar natural conditions, and points out general types of mining plans that have established high recovery ratios with a minimum of mine development and with relatively low accident rates.

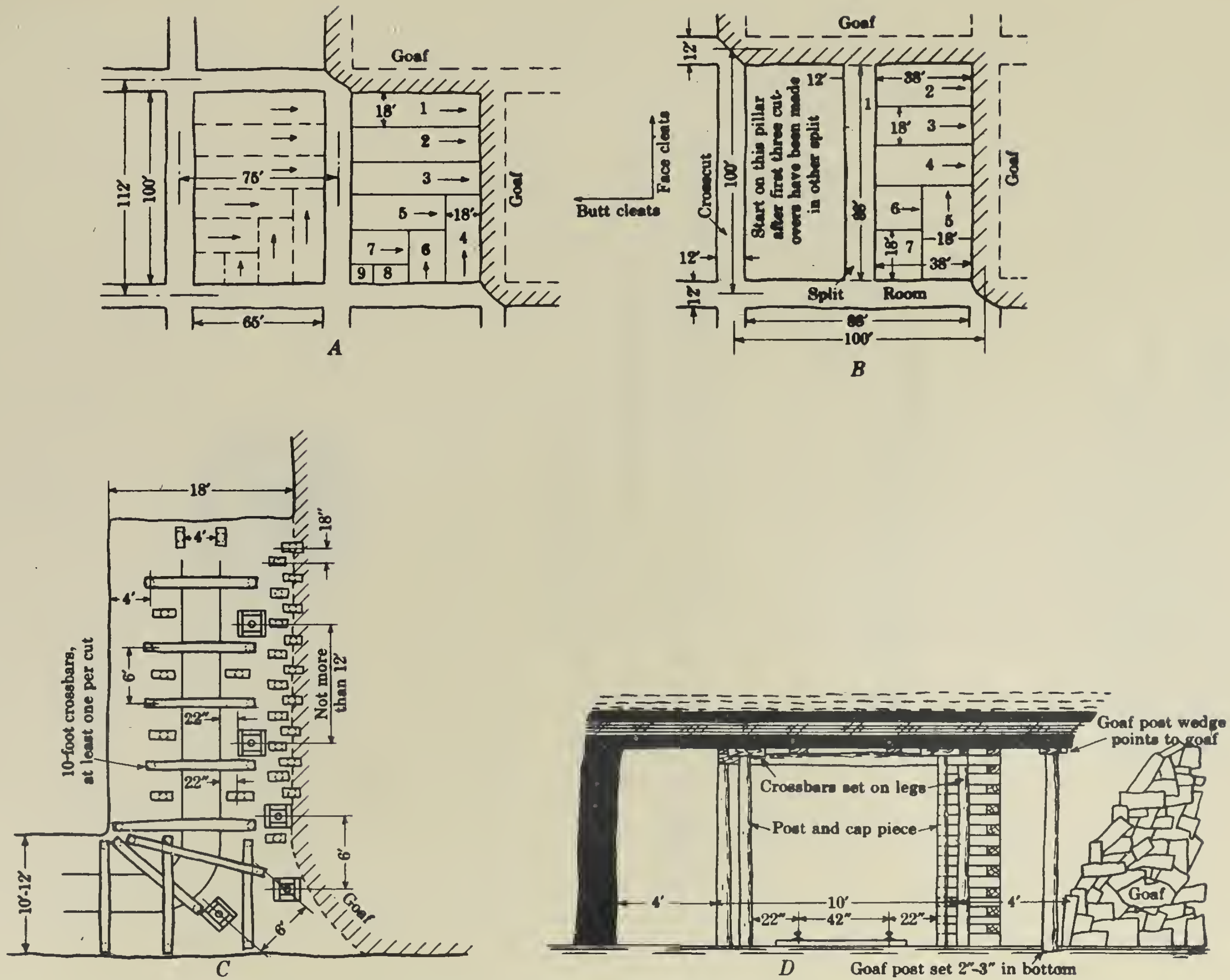

Figure 22.- Sequence of open-end pillar mining and timbering methods as practiced by mines in the coking region of Pennsylvania, group E, type E mines: A, Sequence of extraction of blocks 65 by 100 feet; B, sequence of extraction for blocks 88 feet square; C, plan of timbering in open-end places; D, section across open end showing timbering details.

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UNITED STATES BUREAU OF MINES

John W. Finch, Director

INFORMATION CIRCULAR

A NOTE ON THE USE OF ULTRAVIOLET LAMPS IN MINES FOR RAPID DETECTION OF SCHEELITE IN ORES BY FLUORESCENCE

BY

William O. Vanderburg

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

A NOTE ON THE USE OF ULTRAVIOLET LAMPS IN MINES FOR RAPID DETECTION OF SCHEELITE IN ORES BY FLUORESCENCE[1]

By William C. Vanderburg[2]

INTRODUCTION

The fluorescence of many minerals, when excited by different wave lengths of ultraviolet radiation, has long been recognized as a spectacular laboratory phenomenon. Since the original display in the British Museum, the phenomenon has been used frequently for educational purposes and public demonstrations. Its practical application for the rapid detection of certain minerals in ores and mill products at the zinc mine at Franklin, N.J., has been described.[3]

Due to technical improvements in ultraviolet-radiation apparatus, fluorescence has found a new field of application in the tungsten-mining industry. Although fluorescence of scheelite (calcium tungstate) has been known for a number of years, the practical application of the phenomenon is the outgrowth of a program of cooperative research conducted by Ott F. Heizer, manager of the Nevada-Massachusetts Co., Inc., Mill City, Nev., and Dr. Paul F. Kerr of Columbia University. Special portable lights for fluorescing scheelite have been developed and utilized in underground geological work.

Thanks are due Ott F. Heizer for permission to publish data included in this paper.

APPARATUS FOR PRODUCING ULTRAVIOLET RADIATION

There are several models of ultraviolet lamps on the market that have been designed for specific purposes, but the most satisfactory one for producing fluorescence in scheelite is the so-called "Strong arc." It is similar in design to the outfit originally described by Andrews.[4]

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2 Mining engineer, U.S. Bureau of Mines.

3 Palache, Charles, The Phosphorescence and Fluorescence of Franklin Minerals: Am. Mineral., vol. 13, no. 7, 1928, pp. 330-333.

4 Andrews, W. S., Apparatus for Producing Ultraviolet Radiation: Gen. Elec. Rev., vol. 19, 1916, pp. 317-319.

A lamp of this type may be purchased for $35, or one may be built at a cost of $12 to $35, depending on the refinements of construction. The outfit can be connected to any 110-volt alternating current circuit. A sketch of the lamp and the wiring diagram is shown in figure 1.

The apparatus consists of three principal parts, which are as follows:

1. A small transformer that steps up 110-120 volt alternating current to 4,500 volts.

2. A mica condenser that has a capacity of 0.004 microfarad at 3,500 volts.

3. A spark gap with adjustable and replaceable iron electrodes.

The above parts, with accessories such as switch, wiring, terminal posts, and receptacle for spark gap, are installed in a neat wood box. The outfit weighs about 18 pounds.

The spark gap is enclosed in a cylindrical chamber of bakelite for insulation and for protecting the eyes of the operator.

It is important, when working with the instrument, that the eyes be shielded from the invisible but none the less injurious ultraviolet radiation. Exposure of the eyes to ultraviolet radiation from electric arc lamps or electric welding outfits causes conjunctivitis, which is an inflammation of the muccous membrane covering the lining of the inside of the eyelid and part of the eyeball.[5]

When the connecting cord of the apparatus is plugged into a suitable current outlet and the spark gap adjusted to about 1/8-inch, the instrument is ready for use.

A portable light similar to the one described, but equipped with a quartz projecting lens, is employed at the tungsten properties of the Nevada-Massachusetts Co., Inc. This type of lamp was designed particularly for the detection of scheelite in mining operations.

The high tension disruptive spark between iron terminals produces an intense spark without flame. The spark is particularly rich in radiation in the ultraviolet region of the spectrum - roughly, from 4,270 to 2,100 Ångstrom units. Although the iron arc emits both visible and invisible radiation, the proportion of visible radiation is low enough to permit efficient use without a filter.

APPLICATION OF THE ULTRAVIOLET LAMP IN TUNGSTEN MINING

When the ultraviolet radiation impinges upon certain minerals, the minerals become fluorescent and glow with various colors. Scheelite, the most important

5 Kindall, C. R., Effect of Ultraviolet Rays on the Eyes: Rept. of Investigations 2173, Bureau of Mines, 1920, 2 pp.

tungsten mineral mined in the United States, is capable of absorbing ultraviolet radiation and transforming this radiation into colors of longer wave length in the visible spectrum. Most scheelite fluoresces a light blue under the iron spark. The fluorescence can be seen in a lighted room or in the shade in daylight; however, it shows to best advantage in a dark room, and it is particularly striking in the total darkness of underground workings.

The color of fluorescence is peculiar to the ore examined, and since color is a simple means of identification, it furnished a rapid and reliable method for the qualitative as well as approximate quantitative determination of scheelite in ores and mill products. This method of increasing the sense of vision is advantageous with scheelite because in ordinary light with the unaided eye it is difficult to distinguish scheelite from minerals ordinarily associated with it, such as quartz and calcite.

In concentrating scheelite ores the instrument is useful in estimating the percentages of scheelite in jig, concentrating-table, and magnetic separator products to test the efficiency of the various machines. A sample of the dried material is placed under the spark lamp, and the amount of scheelite in the sample can be approximated by its fluorescence. On the finer-mesh material the fluorescence of scheelite does not show as readily in a lighted room as on the coarser material. A dark room is necessary for the best results. Due to the fact that chemical methods of assying for tungsten are slow and expensive, chemical methods of assaying are seldom, if ever, used in routine mill-control work at tungsten concentrators. Ordinarily, methods based on the specific gravity of the tungsten minerals are employed. In the case of scheelite, the ultraviolet apparatus is more rapid than panning, and, because of the greater area available for exposure, gives a better average.

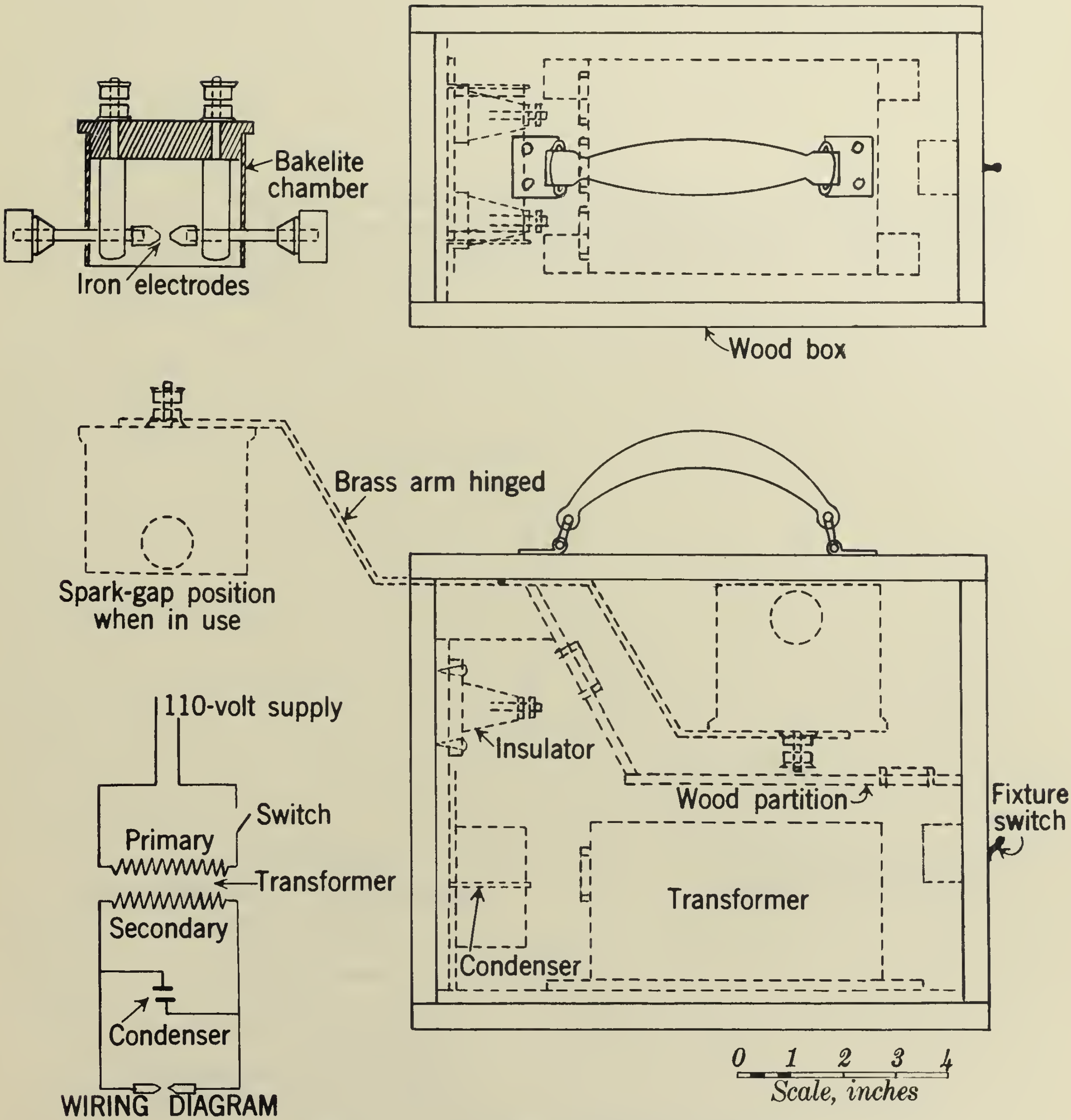

Figure 1.—Ultraviolet radiation apparatus.

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INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

METHANE-INDICATING DETECTORS PROVE DEPENDABLE IN SAMPLING AIR IN ANTHRACITE MINES[1]

By R. D. Currie[2]

The practicability and dependability of two permissible methane-indicating detectors recently developed were proved conclusively by extensive tests in the return airways of all mines operating in the anthracite region in most of which samples and readings were taken with one or the other of the two indicating detectors. These detectors proved dependable and accurate within practical limits, and their general use should be encouraged as an aid to better ventilation, more accurate control of air currents, and greater safety in mines.

DESCRIPTION OF DETECTORS

The U.C.C. and M.S.A. detectors approved by the Bureau of Mines as permissible under Schedule 8B were used in this study.

A complete description of the U.C.C. detector was published in Bureau of Mines Bulletin 331, following a series of tests to determine the practicability of the various detectors approved at that time. The M.S.A. detector was developed and improved after the publication of this bulletin and is therefore not mentioned in the description of the four permissible methane detectors approved before 1930.

The U.C.C. and M.S.A. detectors have the following features in common: They are operated electrically and use the Wheatstone-bridge principle (modified) based on the combustion of methane in contact with an electrically heated platinum filament; they have indicating meters which give readings in direct percentage of methane; and they are portable, are connected with permissible electric cap lamps, and are permissible for use in gassy mines.

Figure 1 shows the details of the U.C.C. detector and permissible electric cap lamp. This detector consists of four parts connected by flexible electric conductors: Detector head A; meter and bridge compartment B; lead-acid storage battery C; and cap-lamp attachment D.

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2/ District engineer, U. S. Bureau of Mines Safety Station, Wilkes-Barre, Pa.

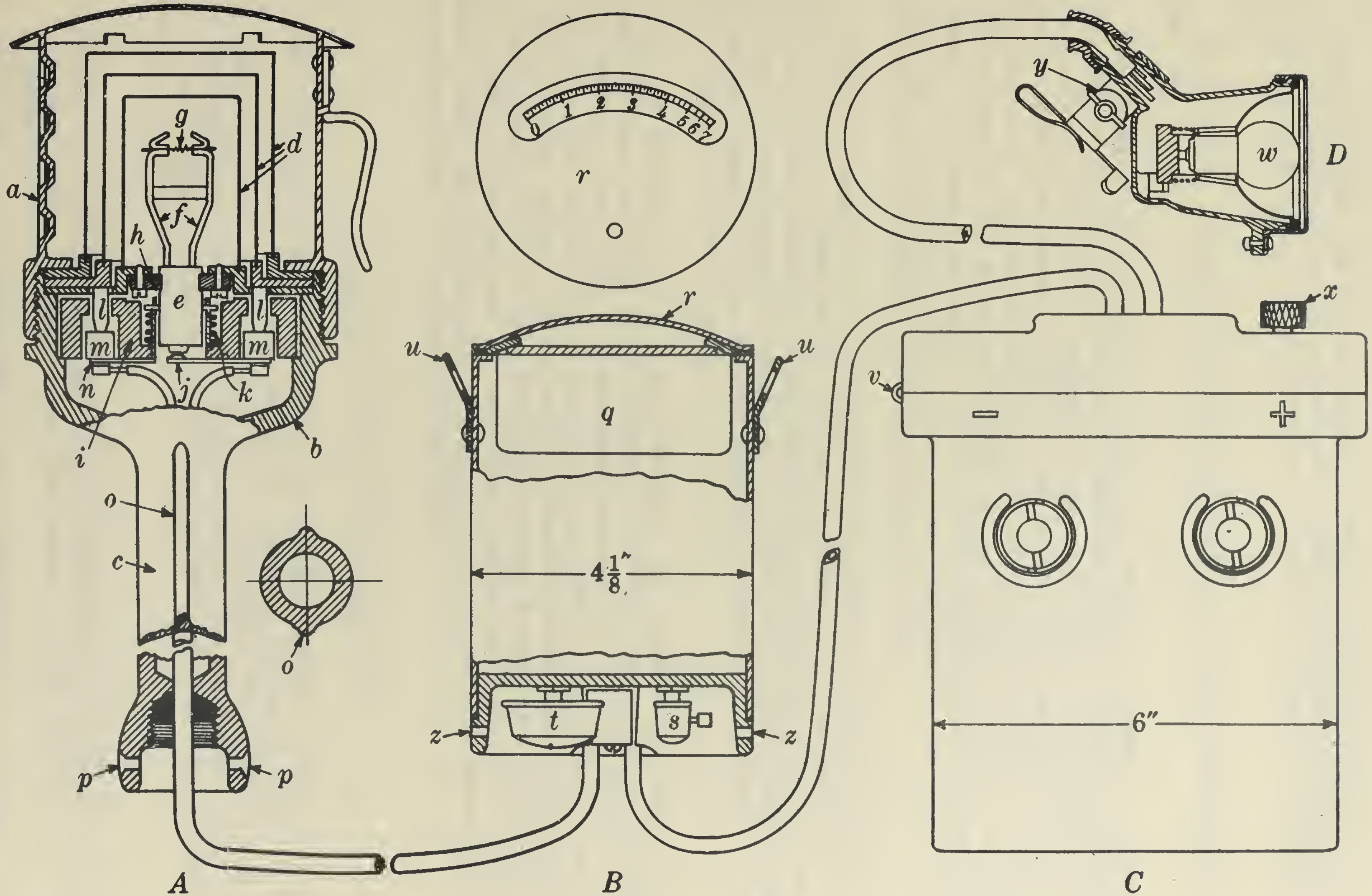


Figure 1.— Details of U.C.C. permissible methane-indicating detector.

Figure 2 shows the M.S.A. detector, which consists of three parts connected by flexible electric conductors: Alkaline storage battery, permissible electric cap lamp, and aluminum box containing filaments, rheostats, and meter.

The electric circuits of these detectors are shown diagrammatically in figure 3.

The fundamental principle on which the detectors operate - change in electrical resistance with change in temperature of a platinum filament - has been known for years; a methane detector based on this principle was invented by G. Leon about 1902[3], but the device never was perfected to the practical stage. A similar device was described in 1914 by G. J. Ralph[3] as an improvement of his earlier device[4], but it proved a failure in detecting firedamp. No commercial adaptation of the principle used in these devices was perfected until the U.C.C. detector was submitted to and approved by the Bureau of Mines in November 1928 under approval 702.

Although based on the same general principle, the M.S.A. detector overcomes two of the weaknesses of the U.C.C. detector - relatively short life of the filament and inability to set the instrument to zero except in strictly fresh air - by considerable variation in electrical circuit and by enclosing the heated filament in the meter compartment where it is exposed to methane only during actual sampling.

As the filament in the M.S.A. detector is exposed to methane only during actual testing, its life is prolonged in inverse proportion to the number of tests made and percentages of methane encountered. It can be set to zero anywhere in a mine by turning on the electric current and allowing the filament to burn out any methane in the small filament compartment. The atmosphere in the compartment is then inert and approximates a condition of fresh air insofar as combustible gas and the consequent temperature of the filament are concerned; these two improvements (especially the latter) are very important to the average user of this type of equipment.

PRACTICAL USE OF METHANE DETECTORS

At the request of the Pennsylvania State Department of Mines the United States Bureau of Mines in September 1932 began a series of tests to determine the methane content of air in the return airways of the mines in the anthracite region. Soon afterward it was decided to use the U.C.C. indicating detector for immediate indications of the percentage of methane present and to determine its practicability.

3/ Ralph, G. J., A Portable Electrical Gas-Detecting Device for Use with Miners' Lamps: Trans. Inst. Min. Eng., vol. 48, 1914-15, pp. 79, 306-309.

4/ Ralph, G. J., The Holmes-Ralph Gas-Detecting Portable Electric Lamp: Trans. Inst. Min. Eng., vol. 42, 1911, p. 201.

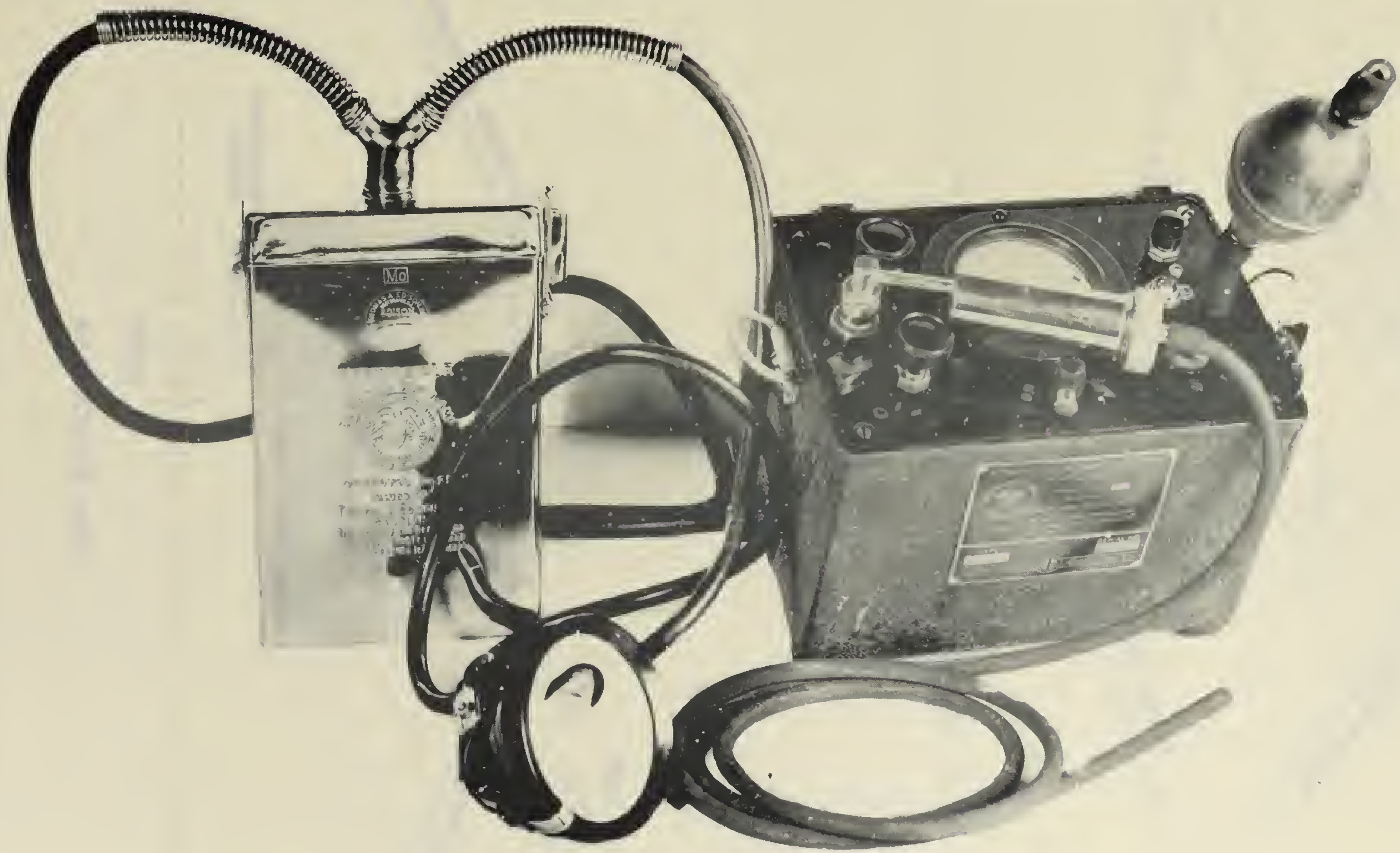
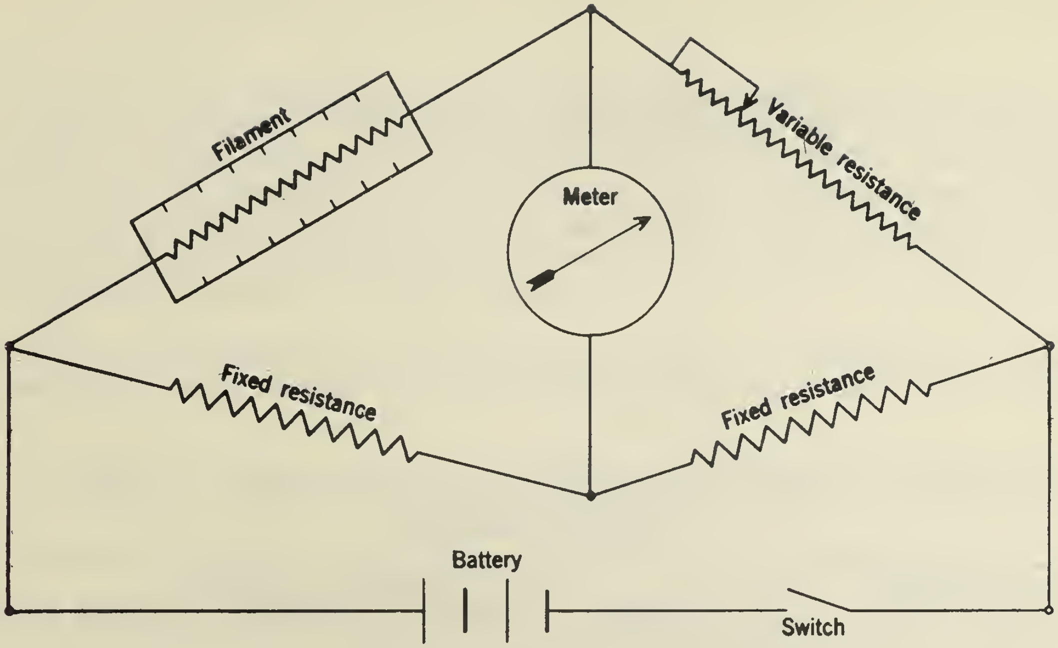
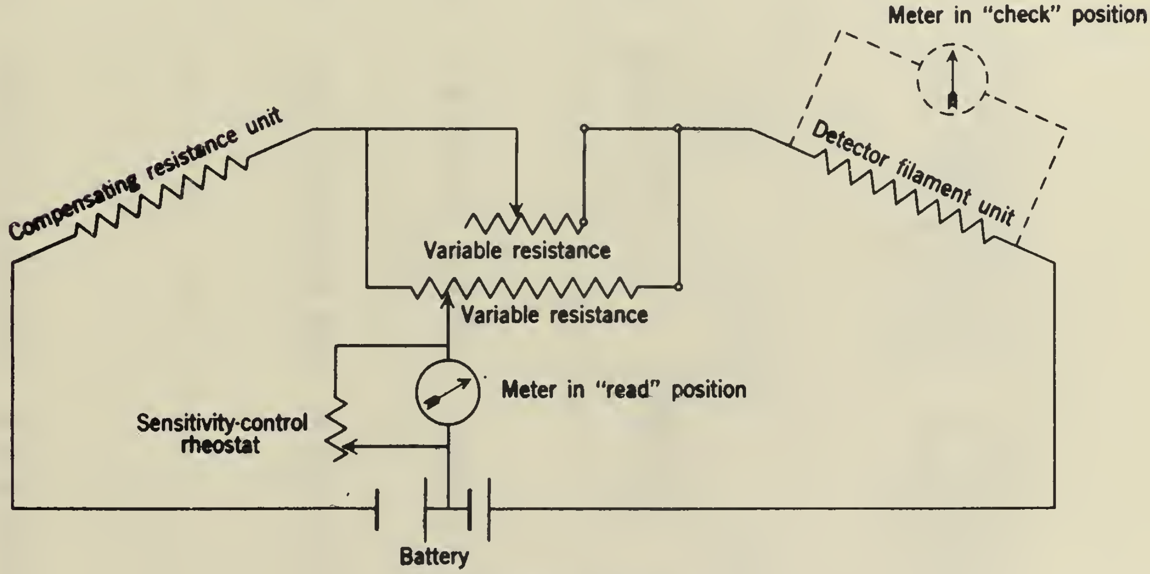


Figure 2.— M.S.A. permissible methane-indicating detector.



U.C.C. DETECTOR CIRCUIT



M.S.A. DETECTOR CIRCUIT

Figure 3.—Electric circuits of the U.C.C. and M.S.A. methane-indicating detectors.

The first large company whose mines were sampled by the Bureau of Mines had equipped its ventilation inspector with a U.C.C. detector, which he carried during the sampling of the airways of all their mines. The results obtained with this instrument, the immediate determinations of methane percentage obtained, and the opportunity for making an exhaustive study of the practicability of the instrument led the Bureau to provide the writer with one for use during the rest of the sampling.

The results of analysis of samples collected by the Bureau of Mines and the readings of the U.C.C. methane-indicating detector are given in table 1.

TABLE 1. - Comparison of U.C.C.-Detector Readings with Analyses by Haldane Method

Type of sample	Methane, percent, by - Analysis	Methane, percent, by - Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Main return	0.12	0.05		0.07	312,000
Do.	.15	.10		.05	312,000
Do.	.30	.20		.10	880,000
Do.	.30	.20		.10	880,000
Do.	.39	.40	0.01		1,250,000
Do.	.38	.40	.02		1,250,000
Do.	.19	.11		.08	253,900
Do.	.19	.11		.08	253,900
Do.	.37	.40	.03		1,190,000
Do.	.37	.40	.03		1,190,000
Do.	.12	.15	.03		577,000
Do.	.15	.15	.00	.00	577,000
Do.	.03	.02		.01	52,500
Do.	.02	.02	.00	.00	52,500
Do.	.28	.32	.04		859,000
Do.	.28	.32	.04		859,000
Do.	.39	.35		.04	852,000
Do.	.38	.35		.03	852,000
Do.	.62	.60		.02	2,700,000
Do.	.62	.60		.02	2,700,000
Do.	.16	.20	.04		480,000
Do.	.15	.20	.05		480,000
Do.	.31	.38	.07		1,485,000
Do.	.32	.38	.06		1,485,000
Do.	.15	.10		.05	598,000
Do.	.15	.10		.05	598,000
Do.	.44	.40		.04	2,675,000
Do.	.46	.40		.06	2,675,000
Do.	.26	.21		.05	882,500
Do.	.24	.21		.03	882,500

TABLE 1. - Comparison of U.C.C.-Detector Readings with Analyses by Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	-	Methane per 24 hours, cubic feet
Main Return	0.86	0.85		0.01	2,080,000
Do.	.88	.85		.03	2,080,000
Do.	.47	.50	0.03		1,979,000
Do.	.50	.50	.00	.00	1,979,000
Do.	.61	.60		.01	2,330,000
Do.	.60	.60	.00	.00	2,330,000
Do.	.37	.32		.05	620,000
Do.	.34	.32		.02	620,000
Do.	.04	.05	.01		90,250
Do.	.04	.05	.01		90,250
Do.	.06	.05		.01	106,500
Do.	.08	.05		.03	106,500
Do.	.09	.08		.01	266,000
Do.	.08	.08	Exact		266,000
Do.	.93	1.05	.12		1,892,000
Do.	.83	1.05	.22		1,892,000
Do.	.02	.00		.02	11,550
Do.	.02	.00		.02	11,550
Do.	.24	.35	.11		258,000
Do.	.24	.35	.11		258,000
Return from split	.34	.60	.26		268,000
Do.	.35	.60	.25		268,000
Do.	.21	.30	.09		161,500
Do.	.22	.30	.08		161,500
Do.	.24	.25	.01		159,500
Do.	.24	.25	.01		159,500
Do.	.11	.00		.11	50,000
Do.	.12	.00		.12	55,000
Do.	.20	.12		.08	59,300
Do.	.19	.12		.07	59,300
Do.	.27	.10		.17	109,800
Do.	.27	.10		.17	109,800
Do.	.29	.25		.04	257,500
Do.	.27	.25		.02	257,500
Do.	.35	.30		.05	143,800
Do.	.33	.30		.03	143,800
Main Return	.21	.10		.11	1,091,000
Do.	.21	.10		.11	1,091,000
Return from split	.33	.32		.01	1,011,000
Do.	.37	.32		.05	1,011,000

TABLE 1. - Comparison of U.C.C.-Detector Readings with Analyses by Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Methane, percent, by - Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Return from split	0.02	0.10	0.08		3,295
Do.	.15	.15	Exact		47,400
Do.	.15	.15	do		47,400
Do.	.25	.21		0.04	160,500
Do.	.24	.21		.03	160,500
Do.	.10	.05		.05	88,600
Do.	.11	.05		.06	88,600
Do.	.13	.15	.02		96,800
Do.	.13	.15	.02		96,800
Main return	.00	.00	Exact		--------
Do.	.00	.00	Exact		--------
Do.	.04	.00		.03	32,300
Do.	.03	.00		.02	24,200
Return from split	20.6	Explosive mixture			--------
Do.	15.4	do			--------
Do.	2.23	2.00		.23	--------
Do.	2.06	2.00		.06	--------
Main return	.30	.30	Exact		254,000
Do.	.26	.30	.04		250,000
Do.	.08	.15	.07		188,946
Do.	.31	.40	.09		493,000
Do.	.11	.15	.04		305,000
Return from split	.00	.02	.02		--------
Do.	.00	.02	.02		--------
Do.	.15	.20	.05		203,000
Do.	.12	.12	Exact		10,500
Do.	.04	.15	.11		50,600
Do.	.00	.00	Exact		--------
Do.	.03	.05	.02		7,590
Do.	.00	.05	.05		--------
Do.	.00	.05	.05		--------
Do.	.03	.03			10,600
Do.	.02	.00		.02	5,450
Main return	.00	.10	.10		--------
Do.	.02	.10	.08		--------
Do.	.14	.30	.16		311,500
Do.	.15	.35	.20		311,500
Do.	.32	.18		.14	372,000
Do.	.30	.19		.11	372,000
Do.	.00	.02	.02		5,640

TABLE 1. - Comparison of U.C.C.-Detector Readings with Analyses by Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	-	Methane per 24 hours, cubic feet
Main return	0.01	0.03	0.02		5,640
Return from split	.01	.00		0.01	--------
Main return	.00	.00	Exact		--------
Do.	.02	.00		.02	17,280
Do.	.04	.00		.04	17,280
Do.	.01	.05	.04		7,540
Do.	.00	.00	Exact		--------
Do.	.00	.00	do		--------
Do.	.01	.00		.01	20,925
Do.	.02	.00		.02	20,925
Do.	.02	.00		.02	47,300
Do.	.04	.00		.04	47,300
Do.	.02	.02	Exact		13,750
Do.	.02	.02	do		13,750
Do.	.01	.00		.01	3,940
Do.	.01	.00		.01	3,940
Do.	.08	.10	.02		43,500
Do.	.07	.10	.03		43,500
Return from split	.00	.00	Exact		--------
Do.	.00	.00	do		--------
Main return	.00	.10	.10		--------
Do.	.00	.10	.10		--------
Return from split	.17	.15		.02	347,000
Do.	.18	.15		.03	347,000
Do.	.00	.08	.08		--------
Do.	.00	.08	.08		--------
Do.	.29	.28		.01	285,000
Do.	.29	.28		.01	285,000
Main return	.01	.05	.04		18,300
Do.	.00	.05	.05		18,300
Return from split	.01	.10	.09		19,100
Do.	.01	.10	.09		19,100
Main return	.00	.00	Exact		--------
Do.	.00	.00	do		--------
Do.	.02	.07	.05		19,290
Do.	.02	.05	.03		19,290
Do.	.03	.05	.02		33,000
Do.	.02	.05	.03		33,000
Do.	.08	.10	.02		167,000
Do.	.09	.10	.01		167,000

TABLE 1. - Comparison of U.C.C.-Detector Readings with Analyses by Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Main return	0.03	0.10	0.07		43,800
Do.	.03	.10	.07		43,800
Do.	.00	.05	.05		35,700
Do.	.03	.05	.02		35,700
Do.	.01	.00		0.01	32,400
Do.	.02	.00		.02	32,400
Face sample	.03	.02		.01	--------
Do.	.04	.02		.01	--------
Return from split	.00	.00	Exact		--------
Do.	.00	.00	do.		--------
Do.	.02	.10	.08		1,615
Do.	.03	.10	.07		1,615
Do.	.04	.00		.03	9,370
Do.	.02	.00		.02	9,370
Do.	.14	.00		.14	51,100
Do.	.14	.00		.14	51,100
Do.	.14	.00		.14	185,000
Do.	.16	.00		.16	185,000
Do.	.08	.00		.08	31,600
Do.	.08	.00		.08	31,600
Main return	.08	.00		.08	111,700
Do.	.06	.00		.06	111,700
Do.	.04	.00		.04	61,500
Do.	.05	.00		.05	61,500
Return from split	.07	.00		.07	41,000
Do.	.06	.00		.06	41,000
Do.	.56	.50		.06	105,500
Do.	.57	.50		.07	105,500
Do.	.35	.40	.05		26,400
Do.	.35	.40	.05		26,400
Do.	.37	.40	.03		157,200
Do.	.38	.40	.02		157,200
Do.	.45	.50	.05		114,200
Do.	.41	.50	.09		114,200
Do.	.11	.10		.01	48,700
Do.	.12	.10		.02	48,700
Main return	.04	.10	.06		32,400
Do.	.03	.10	.07		32,400
Do.	.04	.00		.04	59,900
Do.	.05	.00		.05	59,900

TABLE 1. - Comparison of U.C.C.-Detector Readings with Analyses by Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Return from split	0.04	0.05	0.01		4,960
Do.	.05	.05	Exact		4,960
Do.	.04	.00		0.04	20,500
Do.	.04	.00		.04	20,500
Do.	.11	.10		.01	84,400
Do.	.10	.10	Exact		84,400
Do.	.12	.10		.02	279,800
Do.	.12	.10		.02	279,800
Do.	.04	.00		.04	65,600
Do.	.04	.00		.04	65,600
Do.	.08	.10	.02		84,000
Do.	.09	.10	.01		112,200
Do.	.08	.10	.02		26,780
Do.	.06	.10	.04		26,780
Do.	.06	.00		.06	19,500
Do.	.05	.00		.05	19,500
Do.	.03	.00		.03	14,450
Do.	.02	.00		.02	14,450
Do.	.11	.00		.11	32,900
Do.	.10	.00		.10	32,900
Do.	.02	.10	.08		30,280
Do.	.04	.10	.06		30,280
Do.	.03	.05	.02		23,190
Do.	.07	.10	.03		25,600
Do.	.07	.00		.07	33,200
Do.	.09	.00		.09	33,200
Do.	.02	.10	.08		10,450
Do.	.03	.10	.07		10,450
Do.	.02	.00		.02	5,260
Do.	.02	.00		.02	5,260
Do.	.00	.15	.15		3,345
Do.	.01	.15	.14		3,345
Do.	.03	.10	.07		8,680
Do.	.02	.10	.08		8,680
Do.	.07	.10	.03		48,100
Do.	.10	.10	Exact		48,100
Do.	.14	.10		.04	140,800
Do.	.15	.10		.05	140,800
Do.	.16	.25	.09		121,000
Do.	.17	.25	.08		121,000

TABLE 1. - Comparison of U.C.C.-Detector Readings with Analyses by Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	-	Methane per 24 hours, cubic feet
Return from split	0.06	0.25	0.19		44,000
Do.	.04	.25	.21		44,000
Main return	.04	.15	.11		31,700
Do.	.04	.15	.11		31,700
Return from split	.12	.10		0.02	42,800
Do.	.09	.10	.01		42,800
Do.	.03	.00		.03	11,000
Do.	.03	.00		.03	11,000
Do.	.03	.00		.03	5,920
Do.	.03	.00		.03	5,920
Do.	.02	.00		.02	23,600
Do.	.01	.00		.01	23,600
Do.	.15	.10		.05	159,000
Do.	.16	.10		.06	159,000
Do.	.00	.00	Exact		---------
Do.	.00	.00	do.		---------
Do.	.03	.00		.03	5,250
Do.	.01	.00		.01	5,250
Do.	.18	.22	.04		164,000
Do.	.16	.22	.06		164,000
Do.	.04	.00		.04	31,600
Do.	.03	.00		.03	31,600
Do.	.05	.00		.05	118,000
Do.	.07	.00		.07	118,000
Do.	.00	.00	Exact		4,300
Do.	.01	.00		.01	4,300
Do.	.03	.05	.02		1,660
Do.	.02	.05	.03		1,660
Do.	.03	.00		.03	14,950
Do.	.05	.00		.05	14,950
Main return	.05	.10	.05		54,250
Do.	.04	.10	.06		54,250
Do.	.51	.35		.16	218,000
Do.	.50	.35		.15	218,000
Do.	.13	.05		.08	120,000
Do.	.12	.05		.07	120,000
Do.	.24	.25	.01		270,000
Do.	.26	.25		.01	270,000
Do.	.18	.20	.02		400,000
Do.	.19	.20	.01		400,000

TABLE 1. - Comparison of U.C.C.-Detector Readings with Analyses by Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Main return	0.06	0.10	0.04		99,500
Do.	.09	.10	.01		99,500
Do.	.25	.30	.05		697,000
Do.	.25	.30	.05		697,000
Do.	.12	.25	.13		143,000
Do.	.12	.25	.13		143,000
Do.	.41	.50	.09		685,000
Do.	.22	.20		0.02	400,000
Do.	.22	.20		.02	400,000
Do.	.05	.15	.10		43,300
Do.	.10	.15	.05		121,000
Do.	.03	.05	.02		24,900
Return from split	.00	.00	Exact		--------
Do.	.00	.00	do.		--------
Main return	.16	.00		.16	423,000
Do.	.15	.00		.15	396,000
Face sample	.03	.00		.03	44,500
Do.	.03	.00		.03	44,500
Return from split	.00	.00	Exact		--------
Do.	.00	.00	do.		--------
Do.	.03	.10	.07		2,130
Do.	.05	.10	.05		2,130
Do.	.07	.10	.03		2,290
Do.	.08	.10	.02		2,290
Do.	.06	.13	.07		5,225
Do.	.06	.13	.07		5,225
Do.	.10	.10	Exact		4,980
Do.	.10	.10	do.		4,980
Main return	.06	.20	.14		67,500
Do.	.05	.20	.15		67,500
Do.	.36	.15		.21	687,000
Do.	.35	.15		.20	687,000
Return from split	.20	.20	Exact		34,200
Do.	.20	.20	do.		34,200
Do.	.14	.20	.06		14,700
Do.	.13	.20	.07		14,700
Main return	.07	.20	.13		91,100
Do.	.08	.20	.12		91,100
Do.	.15	.15	Exact		169,900
Do.	.15	.15	do.		169,900

TABLE 1. - Comparison of U.C.C.-Detector Readings with Analyses by Haldane Method (Concluded)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Return from split	0.06	0.15	0.09		1,985
Do.	.06	.15	.09		1,985
Do.	.41	.35		0.06	49,800
Do.	.40	.35		.05	49,800
Do.	.50	.55	.05		372,000
Do.	.35	.30		.05	278,500
Do.	.37	.30		.07	278,500
Do.	.03	.00		.03	7,580
Main return	.20	.15		.05	123,500
Do.	.76	.75		.01	589,000
Do.	.11	.10		.01	475,000
Do.	.15	.10		.05	475,000
Do.	.08	.05		.03	135,000
Do.	.10	.05		.05	135,000
Do.	.19	.10		.09	236,500
Do.	.21	.10		.11	236,500

Number of comparisons. . . . . 323[1]
Number of high readings (+). 132
Number of low readings (-) 152
Number of exact readings 39

Average variation in high readings 0.065
Average variation in low readings. 053
Average variation in all readings, disregarding signs. 051

Maximum percentage methane (by analysis) 2.23
Average percentage methane (by analysis) 15

1 Does not include two samples above the range of the detector.

In January 1933 the writer was provided with an M.S.A. detector which was used in sampling return airways of 79 mines; results of these tests are compared with laboratory analyses in table 2.

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses by Haldane Method

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Return from split	0.20	0.15		0.05	34,200
Do.	.47	.35		.12	97,200
Main return	1.24	1.10		.14	2,564,640
Do.	1.25	1.10		.15	2,564,640
Do.	.25	.26	0.01		193,000
Do.	.24	.26	.02		193,000
Return from split	.14	.15	.01		15,250
Do.	1.50	1.20		.30	1,510,000
Main return	.04	.05	.01		23,350
Do.	.02	.05	.03		23,350
Return from split	.01	.02	.01		720
Do.	.01	.02	.01		720
Main return	.01	.00		.01	6,912
Do.	.01	.00		.01	6,912
Do.	.02	.06	.04		16,900
Do.	.00	.06	.06		16,900
Do.	.02	.00		.02	2,000
Do.	.01	.00		.01	1,000
Do.	.00	.05	.05		--------
Do.	.00	.05	.05		--------
Return from split	.00	.07	.07		--------
Do.	.00	.07	.07		--------
Do.	.06	.10	.04		5,190
Do.	.06	.10	.04		5,190
Do.	.44	.50	.06		31,500
Do.	.47	.50	.03		31,500
Main return	.04	.05	.01		131,000
Do.	.04	.05	.01		131,000
Return from split	.16	.28	.12		266,112
Do.	.17	.28	.11		266,112
Main return	.40	.38		.02	1,040,000
Do.	.40	.38		.02	1,040,000
Do.	.55	.52		.03	1,181,000
Do.	.52	.52	Exact		1,181,000
Do.	.06	.10	.04		102,500
Do.	.04	.10	.06		102,500
Do.	.00	.05	.05		--------
Do.	.00	.05	.05		--------
Do.	.07	.07	Exact		223,800
Do.	.06	.07	.01		223,800

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses By Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Main return	0.42	0.39		0.03	1,149,630
Do.	.40	.39		.01	1,149,630
Do.	.40	.47	0.07		1,035,000
Do.	.41	.44	.03		1,035,000
Do.	.35	.34		.01	692,000
Do.	.36	.34		.02	692,000
Do.	.27	.26		.01	796,000
Do.	.25	.26	.01		796,000
Do.	.04	.05	.01		61,200
Do.	.02	.05	.03		61,200
Do.	.01	.00		.01	2,810
Do.	.00	.00	Exact		--------
Do.	.00	.02	.02		--------
Do.	.00	.02	.02		--------
Do.	.02	.00		.02	33,840
Do.	.00	.00	Exact		--------
Do.	.02	.02	do.		4,960
Do.	.00	.02	.02		--------
Do.	.02	.05	.03		27,500
Do.	.02	.05	.03		27,500
Do.	.38	.60	.22		188,500
Do.	.37	.60	.23		188,500
Return from split	.65	.50		.15	85,000
Do.	.68	.50		.18	85,000
Do.	.64	.45		.19	87,500
Do.	.63	.45		.18	87,500
Do.	.02	.04	.02		1,094
Do.	.02	.04	.02		1,094
Do.	.02	.05	.03		542
Do.	.02	.05	.03		542
Do.	.02	.05	.03		1,345
Do.	.02	.05	.03		1,345
Do.	.03	.02		.01	984
Do.	.02	.02	Exact		984
Do.	.02	.05	.03		1,820
Do.	.02	.05	.03		1,820
Main return	.17	.18	.01		60,000
Do.	.18	.18	Exact		60,000
Return from split	.02	.05	.03		34,900
Do.	.03	.05	.02		34,900

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses By Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Return from split	0.03	0.04	0.01		29,900
Do.	.03	.04	.01		29,900
Do.	.01	.00		0.01	2,540
Do.	.00	.00	Exact		--------
Main return	.02	.00		.02	5,600
Do.	.03	.00		.03	5,600
Do.	.03	.05	.02		4,100
Do.	.01	.05	.04		4,100
Do.	.02	.05	.03		7,260
Do.	.01	.05	.04		7,260
Do.	.02	.09	.07		3,840
Do.	.02	.09	.07		3,840
Do.	.01	.02	.01		4,457
Do.	.02	.02	Exact		4,457
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Do.	.00	.05	.05		--------
Do.	.00	.05	.05		--------
Return from split	.00	.08	.08		--------
Do.	.00	.08	.08		--------
Do.	.00	.05	.05		--------
Do.	.00	.05	.05		--------
Do.	.19	.20	.01		148,853
Do.	.18	.15		.03	87,091
Main return	.09	.05		.04	96,941
Do.	.13	.05		.08	96,941
Return from split	.55	.40		.15	--------
Do.	.57	.40		.17	--------
Main return	.04	.10	.06		8,864
Do.	.02	.10	.08		8,864
Return from split	.19	.14		.05	18,147
Do.	.04	.09	.05		7,354
Do.	.10	.12	.02		5,766
Do.	.02	.10	.08		--------
Main return	.05	.05	Exact		95,644
Do.	.05	.05	do.		95,644
Do.	.11	.10		.01	181,500
Do.	.12	.10		.02	181,500

[illegible]

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses By Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Return from split	0.09	0.08		0.01	135,823
Do.	.09	.08		.01	135,823
Do.	.09	.14	0.05		--------
Do.	.19	.25	.06		70,225
Do.	.12	.07		.05	4,095
Do.	.13	.07		.06	4,095
Main return	.04	.07	.03		143,586
Do.	.04	.07	.03		143,586
Return from split	.05	.04		.01	12,549
Do.	.13	.10		.03	24,124
Do.	.05	.05	Exact		16,236
Do.	.03	.00		.03	--------
Do.	.06	.05		.01	5,654
Do.	.02	.00	.02		2,677
Main return	.00	.00	Exact		--------
Do.	.00	.00	do.		--------
Return from split	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Main return	.00	.01	.01		--------
Do.	.00	.01	.01		--------
Do.	.01	.01	Exact		9,800
Do.	.00	.01	.01		--------
Do.	.02	.00		.02	16,390
Do.	.02	.00		.02	16,390
Return from split	.05	.00		.05	19,500
Do.	.05	.00		.05	19,500
Main return	.05	.00		.05	12,000
Do.	.03	.00		.03	10,790
Do.	.03	.00		.03	26,000
Do.	.03	.00		.03	26,000
Do.	.02	.00		.02	2,160
Do.	.02	.00		.02	2,160
Do.	.00	.00	Exact		--------
Do.	.00	.00	do.		--------
Return from split	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses By Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Main return	0.02	0.05	0.03		8,294
Do.	.01	.05	.04		8,294
Return from split	.00	.00	Exact		--------
Do.	.01	.00		0.01	--------
Do.	.01	.02	.01		318
Do.	.01	.02	.01		318
Do.	.07	.08	.01		4,977
Do.	.05	.08	.03		4,977
Main return	.00	.03	.03		--------
Do.	.00	.03	.03		--------
Face sample	.16	.19	.03		--------
Do.	.14	.12		.02	--------
Main return	.06	.05		.01	21,200
Do.	.06	.05		.01	21,200
Do.	.08	.15	.07		13,450
Do.	.08	.15	.07		13,450
Face sample	.00	.02	.02		--------
Do.	.02	.05	.03		--------
Main return	.03	.02		.01	56,400
Do.	.01	.02	.01		56,400
Face sample	.00	.00	Exact		--------
Do.	.00	.00	do.		--------
Do.	.08	.08	do.		--------
Do.	.04	.00		.04	--------
Main return	.01	.04	.03		1,680
Do.	.01	.04	.03		1,680
Do.	.02	.03	.01		4,860
Do.	.01	.03	.02		4,860
Face sample	.00	.02	.02		--------
Return from split	.00	.03	.03		--------
Do.	.02	.03	.01		3,150
Main return	.02	.02	Exact		4,300
Do.	.02	.02	do.		4,300
Do.	.00	.04	.04		--------
Do.	.00	.04	.04		--------
Return from split	.00	.02	.02		--------
Do.	.00	.02	.02		--------
Do.	.02	.03	.01		162
Do.	.01	.03	.02		162

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses By Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Return from split	0.29	0.30	0.01		232,000
Do.	.31	.30		0.01	232,000
Do.	.13	.15	.02		39,100
Do.	.23	.25	.02		94,400
Do.	.05	.05	Exact		7,800
Do.	.04	.05	.01		7,800
Do.	.16	.15		.01	78,200
Do.	.10	.15	.05		78,200
Do.	.05	.06	.01		26,300
Do.	.05	.06	.01		26,300
Face sample	.12	.13	.01		--------
Do.	.12	.13	.01		--------
Main return	.45	.43		.02	657,000
Do.	.45	.43		.02	657,000
Do.	.18	.20	.02		435,000
Do.	.17	.20	.03		435,000
Return from split	.03	.11	.08		29,900
Do.	.05	.11	.06		29,900
Do.	.10	.14	.04		82,500
Do.	.10	.14	.04		82,500
Main return	.06	.10	.04		79,250
Do.	.05	.10	.05		79,250
Do.	.06	.15	.09		172,900
Do.	.06	.15	.09		172,900
Do.	.03	.04	.01		85,700
Do.	.04	.04	Exact		85,700
Do.	.02	.10	.08		51,800
Do.	.01	.10	.09		51,800
Return from split	.01	.05	.04		650
Do.	.01	.05	.04		650
Do.	.01	.06	.05		1,495
Do.	.01	.06	.05		1,495
Do.	.01	.11	.10		1,535
Do.	.02	.11	.09		1,535
Do.	.33	.33	Exact		381,000
Do.	.33	.33	do.		381,000
Do.	.05	.05	do.		28,400
Do.	.03	.05	.02		28,400
Do.	.05	.15	.10		6,500
Do.	.04	.15	.11		6,500

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses By Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Methane, percent, by - Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Main return	0.03	0.03	Exact		45,410
Return from split	.05	.02		0.03	37,400
Do.	.06	.02		.04	37,400
Main return	.11	.04		.07	238,000
Do.	.12	.04		.08	238,000
Do.	.01	.00		.01	17,500
Do.	.01	.00		.01	17,500
Return from split	.00	.01	0.01		--------
Do.	.01	.01	Exact		173
Do.	.01	.05	.04		--------
Do.	.02	.05	.03		--------
Do.	.02	.01		.01	168
Do.	.01	.01	Exact		168
Do.	.06	.06	do.		13,793
Do.	.06	.06	do.		13,793
Main return	.01	.02	.01		3,970
Do.	.03	.02		.01	3,970
Do.	.02	.05	.03		8,840
Do.	.03	.05	.02		8,840
Return from split	1.51	1.35		.16	26,200
Do.	1.52	1.35		.17	26,200
Do.	.05	.10	.05		--------
Do.	.05	.10	.05		--------
Do.	.05	.10	.05		5,310
Do.	.05	.10	.05		5,310
Do.	.03	.10	.07		1,290
Do.	.02	.10	.08		1,290
Main return	.03	.00		.03	62,400
Do.	.03	.00		.03	62,400
Do.	.02	.05	.03		35,700
Do.	.05	.05	Exact		35,700
Do.	.04	.05	.01		72,400
Do.	.04	.05	.01		72,400
Do.	.02	.10	.08		28,800
Do.	.02	.10	.08		28,800
Do.	.05	.15	.10		110,500
Do.	.06	.15	.09		110,500
Do.	.02	.10	.08		36,000
Do.	.03	.10	.07		36,000
Return from split	.13	.10		.03	--------
Do.	.12	.10		.02	--------

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses By Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Return from split	0.03	0.00		0.03	--------
Do.	.01	.00		.01	--------
Main return	.01	.00		.01	2,900
Do.	.01	.00		.01	2,900
Do.	.22	.15		.07	55,000
Do.	.22	.15		.07	55,000
Do.	.03	.05	0.02		287,239
Do.	.04	.05	.01		287,239
Do.	.04	.05	.01		188,326
Do.	.03	.05	.02		188,326
Do.	.14	.10		.04	115,200
Do.	.13	.10		.03	115,200
Do.	.20	.20	Exact		394,000
Do.	.20	.20	do.		394,000
Do.	.03	.05	.02		44,500
Do.	.03	.04	.01		17,900
Return from split	.02	.02	Exact		6,620
Do.	.02	.02	do.		6,620
Main return	.05	.05	do.		59,700
Do.	.04	.05	.01		6,700
Do.	.00	.02	.02		--------
Do.	.01	.02	.01		--------
Do.	.02	.02	Exact		--------
Do.	.01	.02	.01		--------
Repeat sample	.01	.00		.01	--------
Do.	.01	.00		.01	--------
Main return	.06	.10	.04		71,700
Do.	.06	.10	.04		71,700
Do.	.06	.10	.04		40,500
Do.	.07	.10	.03		47,300
Do.	.08	.15	.07		91,200
Do.	.08	.15	.07		91,200
Return from split	.03	.06	.03		6,550
Do.	.03	.06	.03		6,550
Do.	.02	.02	Exact		3,790
Do.	.00	.02	.02		3,790
Do.	.01	.03	.02		927
Do.	.05	.00		.05	9,161
Do.	.01	.03	.02		--------
Do.	.05	.05	Exact		5,637

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses By Haldane Method (Continued)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	Difference -	Methane per 24 hours, cubic feet
Return from split	0.04	0.00		0.04	9,266
Main return	.27	.16		.11	903,753
Do.	.24	.16		.08	903,753
Do.	.03	.02		.01	2,923
Do.	.05	.02		.03	2,923
Do.	.00	.01	0.01		4,246
Do.	.01	.01	Exact		4,246
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Do.	.04	.00		.04	17,110
Do.	.04	.00		.04	17,110
Return from split	.03	.04	.01		30,142
Do.	.04	.04	Exact		30,142
Do.	.02	.05	.03		2,442
Do.	.02	.05	.03		2,442
Do.	.80	.80	Exact		22,778
Do.	.78	.80	.02		22,778
Main return	.04	.00		.04	12,530
Do.	.02	.00		.02	12,530
Do.	.00	.00	Exact		--------
Do.	.00	.00	do.		--------
Do.	.03	.03	do.		36,450
Do.	.02	.03	.01		36,450
Return from split	.02	.05	.03		13,040
Do.	.02	.05	.03		13,040
Do.	.01	.05	.04		2,407
Do.	.00	.05	.05		2,407
Main return	.00	.00	Exact		--------
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Do.	.01	.00		.01	1,260
Return from split	.00	.03	.03		--------
Do.	.00	.03	.03		--------
Do.	.00	.02	.02		--------
Do.	.00	.02	.02		--------
Main return	.00	.03	.03		--------
Do.	.01	.03	.02		--------
Return from split	.00	.00	Exact		--------
Do.	.00	.00	do.		--------

TABLE 2. - Comparison of M.S.A.-Detector Readings with Analyses By Haldane Method (Concluded)

Type of sample	Methane, percent, by - Analysis	Detector	Difference +	-	Methane per 24 hours, cubic feet
Return from split	0.00	0.00	Exact		--------
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Do.	.00	.00	do.		--------
Main return	.00	.00	do.		--------
Do.	.00	.00	do.		--------

Number of comparisons 364
Number of high readings (+) 194
Number of low readings (-) 98
Number of exact readings 72

Average variation in high readings 0.038
Average variation in low readings 045
Average variation in all readings, disregarding signs 032

Maximum percentage methane (by analysis) 1.52
Average percentage methane (by analysis) 096

Every reasonable effort was made to follow the instructions of the manufacturers in caring for the instruments, including charging and caring for the batteries, except that whenever the opportunity presented itself both instruments were demonstrated to mine officials by gasoline fumes or acetylene gas when no methane was available for the purpose.

Some of the important practical aspects of the results obtained, life of the parts, and reactions of the user are given in table 3 for comparison:

TABLE 3. - Comparison of U.C.C. and M.S.A. methane detectors

	U.C.C. Detector	M.S.A. Detector
Weight and distribution	$10\frac{1}{2}$ pounds; well distributed.	$16\frac{1}{4}$ pounds; poorly distributed.
Maximum weight of any one part	$6\frac{1}{2}$ pounds; battery.	$10\frac{1}{2}$ pounds; meter case.
Accuracy in practice, average variation	0.05	0.032
Practicability for mine use	Good.	Good.
Life of filaments	Relatively short.	Relatively long.
Light from attached cap lamp	Poor.	Good.
Life of battery per charge in practice	5 hours.	5 hours.
Difficulties with batteries	Short life; requires strict care.	Corrosion in "K" model.
Difficulties with other parts	None.	Blocking of orifices in air circuit.

DIFFICULTIES ENCOUNTERED AND HOW CORRECTED

The numerous difficulties that may be encountered through failure to follow the manufacturers instructions for use and care of the detectors are not discussed because it is assumed that these instructions will be followed as carefully by other users of detectors as they were in these tests.

The work presented a few difficulties that undoubtedly caused the readings to vary more than they should have:

M.S.A. detector:
1. Corrosion of terminals and leads.
2. Blocking of orifices.
3. Packing of glass wool in drier tube.
4. Oxidation of rheostat and switches.

U.C.C. detector:
1. Poor illumination from cap lamp.
2. Inability to check zero setting in different mines.
3. Relatively short life of filaments.

These difficulties are overcome readily when the user recognizes them but at first may cause errors in readings that will affect the results.

M.S.A. Detector

1. It was found that the Model "K" Edison cap-lamp batteries used with the detector and with the cap lamps only corrode to a greater extent than the older-type Edison batteries. This difficulty was overcome by thoroughly cleaning the cells, container, and rubber sacks and applying a coat of liquid rubber cement to the tops of the cells and around the edges of the rubber sacks.

2. Blocking of any of the orifices or any other part of the air-sampling circuit may cause a wide variation in methane indication; in practice this did not occur frequently. The difficulty is apparent when the meter shows an indication of methane in strictly fresh air. The air-flow sample is closely controlled by small orifices at the intake and exhaust ends of the sampling circuit, both of which must be clean, but care must be taken not to enlarge the openings in cleaning them.

3. Blocking occurs generally at the bottom of the drier tube where the glass wool packs too tightly on the screen, or calcium chloride dust cakes on the screen; this can be remedied by removing the glass wool occasionally, fluffing it out, and making sure that the screen is clean and the calcium chloride is in good condition.

4. An oxide coating on rheostat and switch contacts affects the flow of electric current through the circuit materially. This difficulty is experienced only when the instrument has been idle for several days; it can be overcome readily by operating the switches and rheostats back and forth several times to polish their surfaces.

U.C.C. Detector

1. The poor illumination from the U.C.C. cap-lamp attachment is inherent in the design and probably can be overcome only by carrying an additional permissible cap lamp or a permissible flashlight.

2. The inability to check zero readings in the mines visited is also inherent in the design of the equipment. Strictly fresh air, or at least air absolutely free from combustible gas, must be available if accurate readings are to be obtained with this apparatus; if this condition is lacking close approximations may be made provided the percentage of methane in certain airways is known and is used as a standard for setting the instrument. In the sampling described in this report neither of these conditions prevailed at times, and some of the results therefore varied rather widely.

3. The life of the filament depends chiefly on the length of time it is used per day and the percentage of methane encountered. As the filament is subjected constantly to increased temperature during travel in return airways containing methane it is advisable when possible to shut off the detector after taking a reading, provided a check location can be found for setting the instrument for the next reading. Spare filaments should be carried dur-

ing inspection trips underground; they can be changed readily, but a new zero setting must be made after a change of filaments.

CONCLUSION

The high degree of accuracy of these permissible methane-indicating detectors, shown in tables 1 and 2 by comparison of analyzed samples with detector readings, and the fact that the indications are immediate and at the location of sampling are distinct advantages to mine officials and mine inspectors.

Every coal mine known to give off explosive gas should be provided with at least one of these detectors to aid the officials in planning for control of ventilation and checking firebosses' and mine-foremen's reports of gas, thus preventing hazardous gas conditions in the mine. Certainly every mine inspector should have one or both of these instruments and should use it enough to become familiar with it. These devices afford the mining man an opportunity to approximate closely the methane content of any part of the mine atmosphere and for practical purposes give data as to explosive gas approximating the value of the information obtained by chemical analysis. However neither of these devices gives information as to oxygen or carbon dioxide content and in this respect both are inferior to the flame safety lamp.

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DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

SAND AND GRAVEL EXCAVATION;

PART 5: MOTOR-TRUCK HAULAGE, CONVEYOR-BELT HAULAGE, PUMPS AND PIPE LINES, BARGES AND TOWBOATS, AERIAL TRAMS

BY

J. R. THOENEN

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

SAND AND GRAVEL EXCAVATION - PART 5: MOTOR-TRUCK HAULAGE, CONVEYOR-BELT HAULAGE, PUMPS AND PIPE LINES, BARGES AND TOWBOATS, AERIAL TRAMS[1]

By J. R. Thoenen[2]

INTRODUCTION

This circular is part 5 of the third paper (entitled "Excavation") of a series summarizing the technical problems involved in the production and preparation of sand and gravel. Part 1 discussed the use of power shovels, draglines, and excavator cranes; part 2, power scrapers, slackline cableway excavators, and hydraulic monitors; part 3, hydraulic, clamshell, ladder, and dipper dredges; and part 4, haulage systems in general, with particular reference to locomotive and car, hoist and car, and remote-control systems. This circular discusses the use of motor trucks, conveyor belts, pumps and pipe lines, barges and towboats, and aerial trams.

MOTOR-TRUCK HAULAGE

During recent years motor trucks have gained considerable ground as a successful competitor of locomotive haulage for quarries and gravel pits. Often where trucks have displaced locomotives and cars the operators readily admit an increase in both labor and total haulage operating costs but point to some other advantage as offsetting the extra expense. In other instances haulage costs have been reduced by the use of trucks. In still others operators, having replaced locomotive haul with trucks, have again returned to the earlier method. Obviously gasoline-driven motor trucks cannot successfully compete with other types of haulage under all conditions. The apparent growth in the number of advocates of truck haulage, however, indicates that it has passed the experimental stage and become a permanent competitor to other types.

Probably the basic reason usually advanced for the increased use of trucks is summed up in the one word "flexibility."

Motor trucks require no rails and ties, although for most economical use it has been found best to provide a well-maintained roadway for them. However, for temporary haulage, trucks can usually service excavating machines

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1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6875."

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without previous road preparation other than the clearing of obstacles and filling of holes. Locomotive haulage requires the laying of tracks and at least some ballasting for even temporary operation. Trucks have more power per ton of load, accelerate quicker, and are capable of shorter time cycles on short hauls. On long hauls the locomotive system will usually handle a larger tonnage per unit of labor and time, even though the speed may be slower. On hauls of several miles over well-ballasted track, however, locomotives may equal or surpass truck speed. Trucks can negotiate steeper grades and sharper curves than locomotives, and these items are of paramount importance in many pits.

Adequate motor-truck equipment operated under good management can supply continuous service to excavator units, and as this type of haulage uses a larger number of independent units delays at the excavator due to the haulage system are liable to be briefer.

One advantage of trucks over other types of haulage is that they can be diverted from pit usage to delivery equipment from plant to consumer when required. Frequently a gravel-pit operator is called upon to fill an order for bank-run material. If his pit is equipped with motor-truck haulage he can usually avoid rehandling material at the plant. Trucks can be loaded by the excavator and sent direct to the consuming point. With other types of pit haulage, bank-run material must usually be unloaded at the plant and reloaded to the delivery equipment.

Trucks require more labor per ton hauled than locomotives. They may consume as much or more fuel per unit handled, and because of the larger number of motor units and the use of rubber instead of steel treads repair and maintenance costs may be greater. Ordinarily, however, for equal capacity the initial investment is lower for truck haul.

Often locomotives may be operated in wet weather when trucks cannot function owing to slippery roads. If a well-maintained roadway has been built between pit and dump this disadvantage largely disappears.

For pits producing large tonnages (several thousand tons daily) locomotives with their greater unit capacity have the advantage over truck haulage because less labor is required for the lower number of haulage units and there is less traffic interference within the pit.

Both types of haulage require careful, adequate, and continuous supervision and maintenance to continue efficient.

Structural Limitations

Motor trucks are built in so many types and sizes that space does not permit enumeration of their details. The drive unit and chassis are usually standardized by each manufacturer, although the standards of one builder may

vary greatly from those of another. Bodies may be built and supplied by the truck builder, but in many instances they are manufactured by separate concerns or by the gravel producer himself. The correlation of body size to tire load is important, and if truck bodies are furnished by concerns other than the truck builder care should be exercised that the load capacity does not exceed the designed tire loading as excessive maintenance costs may then result. Bodies range from 1 to 15 or more tons in capacity and may be arranged to dump to one or both sides or to the rear end. Some are designed to be dumped by gravity, others by mechanism operated by the truck motor, and still others by independent means. They may be constructed of wood or structural steel, and recently aluminum alloys have received considerable commendation by operators who have experimented with them in attempts to decrease the dead load and increase the pay load on the same unit.

Trucks may be operated as single units or may haul one or more trailers, each of which may have an equal or even greater capacity than the truck itself. The addition of trailers, however, is usually attended by a decrease in flexibility, as such units are slower and require extra maneuvering, both at the excavator and the dump; however, the use of one or more trailers has been found economical for long delivery hauls, particularly in western practice. Trailers have usually been discarded for pit to plant service.

Trucks are well-adapted for either short or long pit hauls. However, more trucks or larger cargo capacities are needed with longer hauls, hence initial expense is greater. There is considerable controversy among operators as to the most economical size of truck. Some maintain that a larger number of small trucks is better, while others prefer a smaller number of large trucks. In any event, the truck body should be of such size as to serve the shovel dipper or dragline bucket adequately and should in no case be less than the maximum capacity of one bucket or dipper. It would appear that for most economical operation the same rule should apply to truck bodies as to cars, namely, that the body should be large enough to hold at least three dipper or bucket loads. Such a rule would limit truck haulage to shovels or draglines with bucket capacities of 4 or 5 cubic yards or less. One probable reason for this limitation is the heavy impact the truck springs must absorb in receiving loads from larger excavators. Truck springs are more flexible than those used on railway equipment, and hence they are not designed for such heavy impact. Sometimes, specially designed springs have been used, but this seems to be a doubtful expedient as the impact is then delivered with less cushioning to the axles and bearings which on trucks are of less rugged construction than those on railway equipment.

Most trucks are equipped with the conventional 3 or 4 speeds and reverse, but in some the gear ratios are arranged to provide as many as 9 forward and 3 reverse speeds.

Trucks can be operated successfully on grades up to 15 percent and even steeper for short hauls. They can make nearly right-angled turns of small radius. The author has even observed rear-dump trucks backing from excavator

to dump and moving forward for the empty return. This expedient is, of course, suitable only for hauls of a few hundred feet. The time required to turn around, however, is saved and the length of the haulage cycle is thereby reduced.

Probably the majority of trucks used for pit haulage dump to the rear. This requires the truck to consume valuable time in backing to the dump. Where trucks are equipped to dump to the side this loss of time may frequently be avoided. Side dumping, however, seems to offer greater structural difficulties, and this may account for its less frequent usage.

Pneumatic tires have largely displaced solid tires for both front and rear truck wheels. The drive is usually through the rear wheels, although some operators prefer both front- and rear-axle drive. To increase traction on rear-wheel drives some builders design each rear wheel to carry two tires. Others use four wheels on the same axle. Still others use two rear axles, either or both of which may be driven and may carry 2 or 4 wheels or tires. Wheel and tire design should be correlated to the weight-carrying capacity of the truck body, as most of the load is supported by the rear or drive wheels and tires.

The bodies of trucks are approximately the same size per ton of load and have the same clearance height above the ground as cars. The driver's cab usually extends above the body and being close to one end prevents a clear swing of the excavator bucket or dipper over the body, thus tending to increase the dumping time of the excavator. The bucket or dipper may be swung above the top of the cab, but when it is dumped the impact on the truck body and springs is then increased.

Trucks are built for speeds up to 40 or more miles per hour, although maximum speeds are seldom attained in pit haulage.

Service Equipment Required

Truck haulage systems to be efficient should have facilities for completely servicing and overhauling motors, tires, and dump bodies. This requires repair shops and garages of sufficient capacity to house all equipment. In cold weather trucks are frequently difficult to start, hence garages should be heated. Where heated garages are not available some operators have adopted the practice of completely draining the radiators at the end of the shift and providing a suitable supply of hot water for refilling them in the morning. This practice, however, is open to criticism owing to the tendency of the cylinder to crack under the strain of sudden expansion due to the introduction of hot water in a cold cylinder block.

While trucks can operate over fairly soft surfaces and even through loose sand, time and money are saved if hard-surfaced roads are prepared for them. Many operators have found it economical to build roads of standard highway construction between pit and plant, extending them as close to the

excavator as possible. These roads should be kept in good repair and all holes promptly filled. At some pits graveled roads are built and a smooth surface is maintained by causing one or more trucks to drag behind them a grader made of railroad rail.

Trucks must be serviced regularly for fuel and oil, but this service is usually rendered at the garage during the noon hour or after the shift is finished.

Many operators insist upon a daily washing and mechanical inspection and periodic greasing of all trucks and bodies. At some plants the truck driver is held responsible for the mechanical care of his truck and paid accordingly. At others a chief mechanic is employed whose sole duty is to see that all trucks are properly serviced.

Recently a new type of truck accessory has been put to use.[3] This consists of a so-called "truck-skip," and is used principally for pits employing hand loading.

Capacity

The capacity of truck-haulage systems is largely a matter of organization and management. The very flexibility of truck units makes careful supervision and timing imperative in order that lost time be kept at a minimum. So far there have been few published analyses of truck haulage problems in the technical literature. Most of these have covered long hauls (several miles) from mine to treatment plant or railway siding and are not applicable to pit conditions. Andrew P. Anderson made an analytical study of truck haulage in connection with the construction of concrete highways, and while his study is not directly applicable to gravel-pit haulage much of the material contained in it can be so utilized.[4] Anderson found that on short hauls (less than 1 mile) the main factor in determining haulage capacity was not the road speed of the unit but the length of time the truck must spend while being loaded and dumped. He also found that, given equal road speeds and normal operating conditions, a 2-, 3-, or 4-batch truck does not deliver to the mixer at the pavement as many batches per hour as 2, 3, or 4 single-batch trucks would. Without further analysis this implies that smaller trucks have greater unit capacity than larger trucks, but when it is recalled that a 4-batch truck must discharge one batch to the mixer, then wait until the mixer is ready for another before dumping the second batch, the reason for this is clear in the lost time at the mixer.

From an analysis of Anderson's study of truck haulage in 122 paving jobs it appears that 9 to 10 percent of the total available haulage time was lost because of faulty equipment or faulty operation of the haulage units. This percentage is computed after deduction of all other time losses attributable to causes or equipment other than the haulage unit itself. Moreover, it does

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3/ Rock Products, Quarry Operator Invents Novel Truck-skip: June 1934, p. 51.
4/ Anderson, A. P., Truck Operation and Production in Concrete Paving Work: Public Roads, vol. 11, no 12, February 1931, p. 247.

not include the time required to load and dump trucks. The 10-percent loss may then be assessed to factors avoidable with well-maintained trucks operated under perfect organization. Since it is not possible for any truck-haulage system to operate with 100-percent efficiency it is reasonable to assume that an average time loss of 10 percent due to the equipment and operation of any truck haulage system may be expected and that variations up or down will result from the degree of efficiency of equipment maintenance and operation.

As pointed out in previous parts of this paper no excavating unit can be expected to operate indefinitely at theoretical capacity, and the working capacities were computed by deducting various percentages from the theoretical. In actual operation excavation equipment does not function in a straight-line curve represented by a percentage of theoretical capacity but in a more or less irregular curve presenting numerous peaks and valleys, the average of which will approach the straight-line curve of working capacity. The capacity of any haulage unit will be found to follow the same rules, especially with such a flexible system as truck haulage. Then, if the peak of the excavator curve coincides with the peak of the haulage curve highest combined efficiency results. Conversely, when the valleys of both curves coincide lowest efficiency results. The haulage unit must be designed to serve the excavator at its peak operation and therefore will have excess units available during less productive periods. These excess units in the circuit must inevitably cause a certain amount of time loss while waiting to be loaded. The time lost in this way cannot be attributed directly to either the excavator or the truck and therefore must be added to the 10 percent assessed to the truck system itself. The amount of loss from this cause will vary from day to day, and even from moment to moment, on any job, depending upon local conditions. No published studies are available from which to evaluate this loss, but from the author's observations efficient all-around service may require the time lost to equal the time required to load one truck. On that basis, the second truck in line would arrive at the excavator just after the first one had started loading. The time consumed by each truck at the excavator would then be twice the actual loading time.

Frequently, time is also lost at the dump with truck haulage. This loss depends largely upon the accommodations provided for unloading. In some instances trucks are run over a pit or hopper and dumped without stopping, but ordinarily the truck comes to a complete stop and stands still until the load is discharged. With trucks designed for side dumping the delay at the dump is usually no more than the actual time required to dump. With rear-dumping trucks further time is usually lost in backing the truck to the dump. Trucks equipped with gravity-dump bodies are frequently difficult to dump owing to the maldistribution of the load by the excavator. The total time required to maneuver and dump the truck will depend entirely upon local conditions. Any truck in good condition should discharge a load of gravel in 20 seconds or less, and maneuvering to a dumping position should not require more than 40 seconds, making a maximum dumping time of 60 seconds. This assumption of the author is arbitrary but it is his opinion that the consumption of more

than 60 seconds by a motor truck at the dump is cause for serious study.

Besides the waiting time at the excavator, there may be corresponding time lost at the dump. This is largely eliminated if the dump is so arranged that the truck can discharge its load without stopping. Where the dumping hopper is too small to accommodate more than one truckload at a time, delay caused by waiting is apt to be unduly large. With the best arrangements there is apt to be some waiting loss due to divergence or coincidence of peaks and valleys in the operating curves of trucks and dump mechanism. For calculation, however, the author feels that one period of loading time at the excavator should be ample to cover the average loss there and include the average loss from a similar cause at the dump, hence no delay factor is used at the dump other than the estimate for maneuvering and dumping.

The truck should, of course, spend the greatest portion of the available hauling time in actual travel between excavator and dump. The empty return distance may vary somewhat from the loaded haul, and the return speed will also vary from the loaded speed, therefore the round-trip speed should be computed by dividing the actual distance covered in the complete cycle by the actual time traveling.

The speed of trucks will be influenced by many factors, among which may be mentioned the mechanical condition of the truck itself; the grade, curvature, and surface condition of the roadway; the amount of traffic interference; whether all units are capable of maintaining the same speed; and the length of the haul. With respect to truck speed Anderson concludes from his study:

First, that with good roads, proper equipment in good condition, freedom from traffic interference, and able management fairly high hauling speeds can be maintained rather consistently.

Second, that with any one of these requirements lacking the uniform maintenance of high speeds is very difficult and probably impossible.

Third, that a high speed is ordinarily of little productive value unless it can be maintained fairly consistently and includes all the hauling units.

Without doubt, these conclusions can be applied equally to pit haulage.

Anderson found that round-trip truck speeds ranged from 6 to 30 miles per hour and averaged 18 miles per hour. An analysis of his table comparing loaded speed with empty return speed shows that the return speed ranged from 4.5 to 38 percent faster than the loaded speed on normally loaded trucks and up to 89 percent faster when overloads were carried on light trucks.

He found further that, other conditions being equal, the average round-trip speed of trucks increased almost invariably with an increased hauling distance but not in computable ratio.

Calculation of the capacity of motor-truck haulage systems involves:

1. The loading speed of the excavator served.

2. The length of round-trip haul.

3. The average round-trip speed.

4. The time consumed in loading and dumping.

5. The time lost from other causes.

6. The cargo capacity of the truck unit.

7. The number of trucks.

From these elements the following formula is borrowed from Anderson's paper, but the factors represented by the components are changed to represent pit haulage.

$$N = \frac{L}{Smt} + \frac{T}{mt},$$

in which

N = number of trucks required;

L = round-trip length of haul, in feet;

S = average round-trip speed, in feet per minute;

T = total time used by each truck in loading, dumping, maneuvering, and waiting on each round trip, in minutes;

t = loading speed of the excavator, in minutes per ton at working capacity;

m = number of tons carried by each truck.

Example 9

Assume an operator has a 1-1/4 cubic yard shovel digging loose gravel which he wishes to serve with trucks. His round-trip haul is 1,500 feet, and road conditions are such that his trucks can average 15 miles per hour while running. It is further assumed that the trucks will lose 10 percent of their available hauling time for various causes and that they will require 1 minute to back and dump at the plant.

From table 9 in part 1 of this paper it is found that the maximum working capacity of 1-1/4-cubic yard shovel is 206 tons per hour.

The loading speed of the shovel is then

$$\frac{20}{206} = 0.291 \text{ minute per ton.}$$

Assuming the gravel weighs 3,000 pounds per cubic yard, each bucket will deliver 1.875 tons. A 5-cubic yard truck will carry 7.5 tons and will be loaded by 4 dipper loads. (Variation in dipper loading is accounted for in calculating shovel working capacity, hence each dipper is here calculated as full.)

The time required to load one truck will then be

$$0.291 \times 7.5 = 2.18 \text{ minutes.}$$

Since 15 miles per hour equals 1,320 feet per minute, the running time of the truck will then be

$$\frac{1500}{1320} = 1.136 \text{ minutes.}$$

The truck cycle, exclusive of lost time, will be as follows:

	Minutes
Loading	2.18
Waiting at shovel	2.18
Running	1.136
Dumping	1.00
Total	6.496

Since it was assumed that there would be a 10-percent loss of time for various causes this 6.496 minutes represents 90 percent of the actual cycle; hence the actual cycle equals

$$\frac{6.496}{0.9} = 7.218 \text{ minutes.}$$

The standstill time plus lost time of the truck per trip is then

$$7.218 - 1.136 = 6.082 \text{ minutes.}$$

Substituting in the formula,

$$N = \frac{L}{Smt} + \frac{T}{mt} = \frac{1500}{1320 \times 7.5 \times 0.291} + \frac{6.082}{7.5 \times 0.291} =$$

$$0.52 + 2.79 = 3.31 \text{ trucks.}$$

To check:

	Minutes
Loading time ----------	2.18
Waiting time ----------	2.18
Running time ----------	1.136
Dumping time ----------	1.00
Lost time -------------	.722
Total cycle --------	7.218

$\frac{60}{7.218}$ = 8.31 trips per truck per hour;

8.31 x 7.5 = 62.3 tons per truck per hour; and

$\frac{206}{62.3}$ = 3.31 trucks required.

With such a short haul it is doubtful if the wait at the shovel need be as great as is assumed in this case, and probably 3 trucks would be sufficient, in continuous operation, especially as the maximum loading capacity of the shovel was used in the computation. However, trucks, like all other kinds of machinery, wear out and require repairs, hence an extra truck as standby equipment is indicated, making 4 trucks.

Example 10

Assume that pit conditions are the same as in example 6 under car and locomotive haulage in part 4 of this paper and that it is desired to substitute truck haulage for the locomotive haulage calculated there.

With the same haulage route the total round-trip haulage distance will be 8,300 feet. At an average truck speed of 15 miles per hour the speed per minute is 1,320 feet. The two 1-1/2 cubic yard shovels will have a combined maximum working capacity of 530 tons hourly and a minimum of 280 tons. In computing locomotive haulage, however, the average was used, or 400 tons hourly, so the same loading rate will be used here. The combined rate is then

$$\frac{60}{400} = 0.15 \text{ minute per ton.}$$

However, the two shovels cannot load the same truck at the same time, hence the loading rate per shovel must be used. Thus,

$$\frac{60}{200} = 0.3 \text{ minute per ton.}$$

It is assumed that each truck can haul a pay load of 6 cubic yards or 9 tons. Then,

	Minutes
Loading time for each truck, 9 x 0.3 -------------------	2.7
Waiting time ----------------------	2.7
Running time, $\frac{8,300}{1,320}$ -----------------	6.29
Dumping time ----------------------	1.00
Operating time --------------------	12.69
Lost time (1/9) -------------------	1.41
Total cycle -----------------------	14.10

The standstill plus lost time will then be

$$14.1 - 6.29 = 7.81 \text{ minutes.}$$

Substituting in the formula,

$$N = \frac{8300}{1320 \times 9 \times 0.3} + \frac{7.81}{9 \times 0.3} = 2.33 + 2.89 = 5.22 \text{ trucks}$$

required for each shovel or, say, 12 trucks in continuous operation.

To check:

Truck cycle = 14.1 minutes;

$\frac{60}{14.1}$ = 4.25 trips per hour per truck;

4.25 x 9 = 38.3 tons per hour per truck;

$\frac{400}{38.3}$ = 10.44 trucks required.

If 12 trucks are used the capacity of the haulage unit would be 2,760 tons in 6 hours or 360 tons more than the 2,400 required.

CONVEYOR-BELT HAULAGE

Belts have been used for years as a means of conveyance between plant units within a single structure, between plant buildings, or between other fixed points. The introduction of the tripper and the shuttle conveyor permitted collection from, or discharge to, more than one point by a single belt. The various points, however, were invariably in a straight line coinciding with the axis of the belt. Application of sectionalized construction to conveyors increased flexibility but still confined operation to straight lines. The belt conveyor was considered permanent equipment, although it was capable of extension by the various devices cited. With the introduction of light-weight, easily handled sections the belt conveyor forced its recognition as a potential competitor for other types of pit haulage. It has been so used by a number of gravel-pit operators with successful results.

The field conveyor is simply an ordinary conveyor redesigned along semiportable lines and adapted to field use. The load and return sections of the belt are carried on rigid structural-steel or alloy frames built in unit sections designed for easy longitudinal extension or lateral movement. The belt itself is in easily connected lengths to fit the added sections.

Field conveyors must still operate in straight lines, but flexibility is provided by sectionalized units. They have been adapted to field haulage in various ways and to fulfilling various functions. They may constitute the sole connecting link between excavator and plant. In so doing they may receive material from the excavator and deliver it direct to the treatment plant either above, below, or level with the pit floor. The conveyors must be of light weight and sectionalized to provide facility of movement in following the progress of the excavator. The plan of excavation may require the excavator to follow a circular or elliptical path or to move in a straight line away from the plant. With a curved working face the conveyor must be constructed in a single unit with one end stationary at the dump and the other moving radially with the excavator, or it must comprise two or more sections working together.

Any pit with a curved working face depending upon a single unit radial conveyor for haulage must soon cover such an extended area that it is uneconomical to move the entire conveyor for each move of the excavator. If the plan of excavation extends in a straight line from the plant it is necessary merely to add successive sections to the conveyor to follow the excavator. However, even with such a plan a distance will finally be reached where it is more economical to displace most of the sectionalized conveyor with permanent construction. With either excavation plan a point is reached at which part of the pit conveyor must be of permanent construction served by a shorter movable unit.

The short movable unit may be a sectionalized conveyor, or it may be an entirely different type of haulage system, such as a fleet of motor trucks or cars and locomotives, or even a pump and pipe line. The belt conveyor then becomes a main haulageway fed by one or more flexible units.

The sectionalized pit conveyor may be so arranged that it follows the excavator, taking material from it and delivering it to another type of main haulage. It then becomes the flexible feeder for a permanent haulage system, such as an aerial tramway, or even a car and locomotive system..

The field conveyor may also be a permanent structure between two fixed points in the pit; at one point material is delivered to it from the excavator by a more flexible type of haulage and at the other the conveyor discharges to a third type of haulage which in turn delivers to the plant. Such a scheme might include truck haulage from the shovel to a centrally located conveyor which delivers to an engine plane for hoisting or lowering to the plant.

While the pit conveyor is restricted to straight-line delivery it is thus capable of considerable flexibility when used with other haulage systems.

Pit conveyors are restricted to use in dry pits or bank deposits. They are not suitable for haulage from wet pits unless some dewatering device is used. They are capable of handling small or large tonnages and, when properly installed, of delivering a continuous flow of material to the plant which is advantageous to efficient plant operation. They require comparatively little power, and as they are fully mechanized are practically automatic in their operation. They require maintenance labor only, and when they are properly installed and not overloaded operating costs are low. Initial and installation costs may be comparatively high, and the cost of moving and realining a long conveyor is expensive.

Structural Limitations

Belt conveyors are endless belts running over supporting idlers and driven through arc contact with one or more driven pulleys. The load is conveyed on the belt itself, hence the belt must be horizontal in cross-section if flat, or if troughed, the edges must be at the same elevation, otherwise the load will spill off the low side. In this position the belt is flexible vertically but not horizontally, hence while it can negotiate vertical curves it cannot operate around horizontal curves. Therefore belt conveyors can carry loads on the level or up or down inclines, but they cannot deviate horizontally from a straight line between terminals.

The belt may be flat or troughed in cross-section, but, as a flat belt offers little resistance to spillage for particles of the load heaped on the center of the belt, troughed cross-sections are preferred for sand and gravel transportation. Troughed belts have about twice the capacity of flat belts

of equal width and speed. The shape of the trough is governed by the position of the idler rolls supporting the belt and varies with the experience of manufacturers and operators. The supporting idlers are constructed so that one or more level rolls carry the center of the belt, and the edges are raised to form the trough by means of one or more troughing rolls on each side which rotate around axles at an angle above the horizontal. The total troughing angle seldom exceeds 30°. The full curvature may be accomplished in a single angle on each side, in which case the angle is usually 20°, or partly in one roll set at a low angle and the balance by second or third rolls at higher angles. The usual arrangement is 15° on the first and 15° more on the second. The details of idler design are highly perfected, with many variations suited to the handling of different classes of material. If the troughing angle is too sharp, excessive stress and wear are put on the belt and it tends to crack at the line of bend; therefore, troughed belts must be thick enough to prevent cracking and may require more plies than flat belts. However, this is seldom the case; for example, a flat belt must be thick enough to prevent excessive sag between idlers. The design must be governed by local conditions. Table 58 gives the minimum and maximum plies required for troughed belts.

TABLE 58.- Minimum and maximum plies for troughed belts

Belt width, inches	Minimum plies required to support load			Maximum plies for troughing
	Sand	Gravel	Boulders	
12	3	4		4
14	3	5		5
16	3	5		5
18	4	5	6	5
20	4	5	6	6
24	4	6	7	6
30	5	6	7	8
36	5	6	8	9
42	6	7	8	10
48	6	7	9	12
54	7	8	10	13
60	7	8	10	14

While a certain amount of vertical curvature is permissible in a conveyor belt it should be kept at a minimum and permitted only through arcs of large radius. Conveyor belts when running are under a tension which varies with the load. If a belt is running horizontally at the loading point and must then ascend a grade the change from the level to the inclined section must be a curve of such radius that the weight of the empty belt holds it on the rolls in spite of the tension required. The maximum adverse conditions requiring curves of the longest radius occur with conveyors having a profile in which there is a long level section carrying a

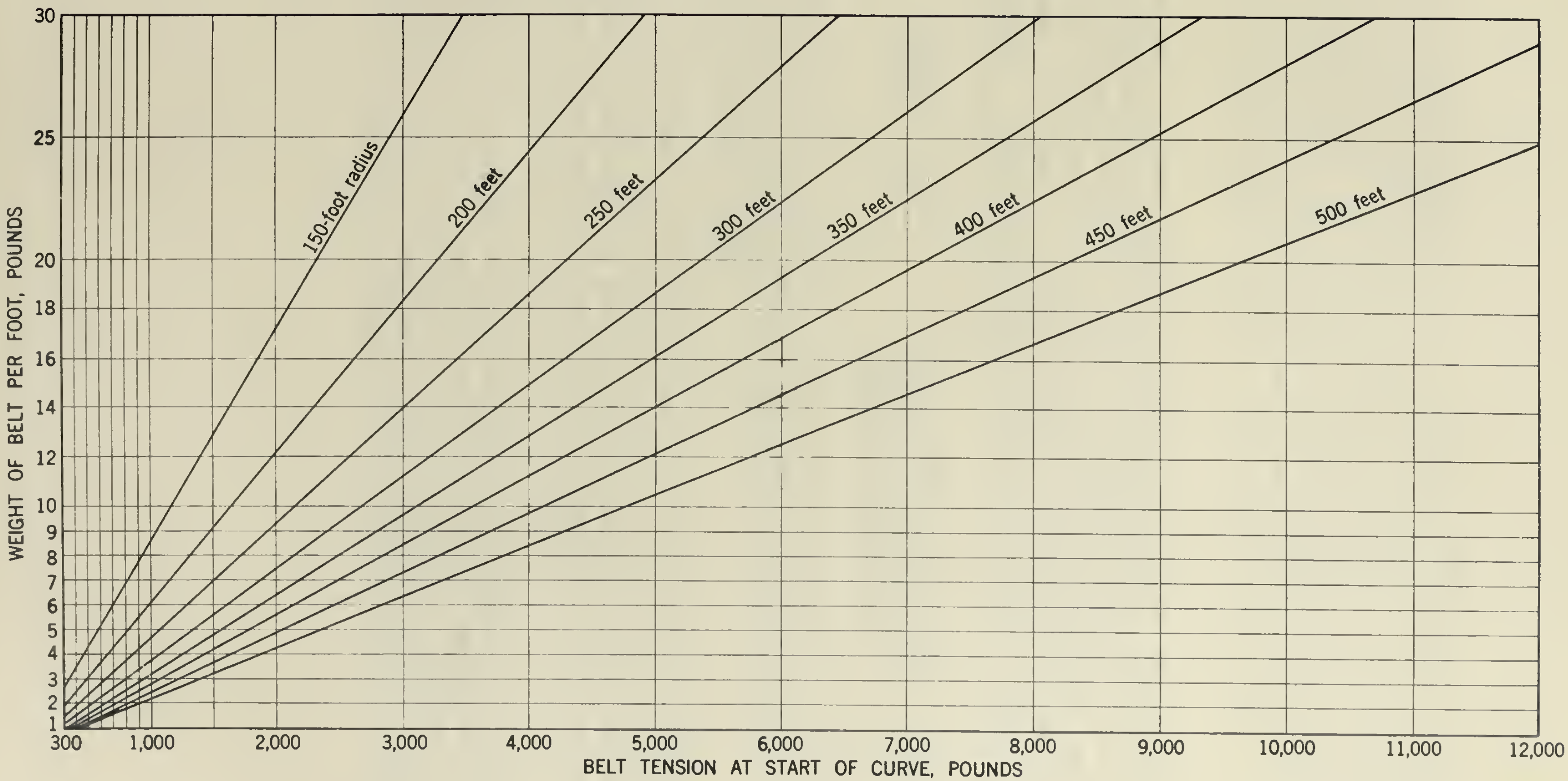


Figure 19.—Chart showing radius for vertical curves in conveyor belts for various belt weights and tensions.

heavy load, followed by a steep incline. The heavy load on the flat portion of the belt causes heavy tension, and if the curve is too sharp the empty belt will be lifted off the troughing idlers at the curve. Determination of the exact curvature required for specific conditions involves the use of complicated mathematical solutions of catenary curves and is a function of conveyor design that should be left to engineers experienced along that line. Figure 19 shows the approximate radius required for belts of various weights operating under various tensions. The radius should never be less than 150 feet.

The maximum angle of inclination over which a conveyor may operate successfully depends upon the angle of repose of the material handled. It will vary with different kinds and sizes of particles making up the load, being greater for fine powders, cubical or rough-edged particles, and damp material and less for lumps, rounded particles, and either thoroughly dry or very wet material. The following are maximum permissible angles of interest to the sand and gravel producer.

Sand (damp)------20°	Gravel, bank run (moist) ---20°
Sand (dry) ------15°	Gravel, unscreened (dry) ---18°
	Gravel, screened (dry) ---15°

Steeper angles have been employed under unusual conditions, but those given should not be exceeded ordinarily. Inclined conveyors should be provided with some means of preventing the belt from running backward if accidentally stopped while under load.

The size of individual pieces which can be carried by a pit conveyor depends upon the width and inclination of the belt, the method of loading, and whether lumps are screened or mixed with fine material. The range of maximum-sized lumps screened or mixed with finer material that can be carried for various widths of belt is shown in table 59.

Field conveyors which must be shifted frequently require a level or smoothly inclined pit floor, otherwise inequalities must be graded or bridged to provide a smooth running belt. Field conveyors of more or less permanent construction can be installed to span existing humps or depressions in the floor which do not require excessive vertical curvature, as discussed previously.

TABLE 59.- Recommended maximum size of lumps for belt-conveyor haulage

Belt width, inches	Lumps uniformly sized, no fines, inches	Lumps in unsized feed, inches
12	1.5 - 2.0	2.0 - 4.0
14	2.0 - 2.5	2.5 - 4.0
16	2.5 - 3.0	3 - 5
18	3.0 - 4.0	4 - 6
20	4.0 - 4.5	5 - 7
24	4.5 - 5.0	6 - 10
30	6 - 7	10 - 14
36	7 - 9	12 - 18
42	8 - 11	14 - 20
48	10 - 14	16 - 24
54	11 - 15	20 - 28
60	12 - 16	24 - 30

The spacing of idler rolls depends upon the weight per cubic foot of the material carried and the width of the belt. Table 60 shows the recommended range.

TABLE 60.- Spacing for carrying rolls, feet1/

Width of belt, inches	Weight of material, pounds per cubic foot			
	10 to 40	40 to 75	75 to 120	120 to 150
12	5.5	5.0 - 5.5	5.0	4.5 - 5.0
14	5.5	5.0 - 5.5	5.0	4.5 - 4.75
16	5.5	5.0 - 5.5	5.0	4.5 - 4.75
18	5.0 - 5.5	4.5 - 5.5	4.5 - 5.0	4.0 - 4.75
20	5.0 - 5.5	4.5 - 5.5	4.5 - 5.0	4.0 - 4.75
24	4.5 - 5.5	4.5 - 5.5	4.0 - 5.0	3.5 - 4.75
30	4.0 - 5.0	4.0 - 5.0	3.5 - 4.5	3.5 - 4.25
36	4.0 - 5.0	3.5 - 5.0	3.5 - 4.5	3.0 - 4.25
42	4.0 - 4.5	3.5 - 4.5	3.5 - 4.0	3.0 - 3.75
48	3.5 - 4.0	3.5 - 4.0	3.0 - 3.5	3.0 - 3.5
54	3.5 - 4.0	3.5 - 4.0	2.75 - 3.5	2.5 - 3.5
60	3.5 - 4.0	3.5 - 4.0	2.75 - 3.5	2.0 - 3.5

1/ Return rolls can be 8 to 10 feet apart.

Conveyor belts are built up of a number of thicknesses or "plies" of canvas or "duck", impregnated and covered with rubber. The maximum length of the conveyor depends upon the strength of the belt or its resistance to tension, which in turn depends on the number of plies used in its construction and the weight of the duck. The weight of the duck is calibrated by the

weight, in ounces per yard, of material 42 inches wide. Canvas of this width commonly used for conveyor-belt construction weighs 28, 32, and 36 ounces per yard; 42-ounce duck is sometimes used for extra-heavy belt. The permissible working stress for each ply per inch of belt width, as recommended by manufacturers, is given in table 61.

TABLE 61.- Maximum working stress per ply, per inch of belt width

Weight of duck, ounces	Working stress, pounds 1/
28	20 - 25
32	23 - 28
36	25 - 30
42	35 - 40

1/ The reason for tabulating a range of working stress is due to its application. The stress applied by counterweights can be definitely controlled and, hence, kept to a minimum. If screw take-ups are used it is difficult to measure and control the stress, hence stronger belts are needed.

The tensional strength of a conveyor belt is therefore proportional to the number of plies of duck used in its construction. This strength cannot be built up indefinitely, however, simply by the addition of extra plies without other complications being encountered. Additional thickness stiffens a belt and reduces its flexibility for troughing and bending around drive pulleys. The maximum plies permissible for proper troughing flexibility were given in table 58. Table 62 gives the minimum recommended pulley diameters for various numbers of plies.

TABLE 62.- Minimum pulley diameters, in inches

Number of plies	Head and drive pulley	Tail, take-up, and snub pulleys 1/
3	15	10
4	20	12
5	24	16
6	30	18
7	36	20
8	42	24
9	48	28
10	54	30
11	54	36
12	60	36
13	66	42
14	72	42

1/ Bend pulleys may be 8 to 16 inches in diameter for any number of plies because of the small arc of contact.

The number of plies contained in standard conveyor belts, as listed in manufacturers' catalogs, is given in table 63.

TABLE 63.- Number of plies of duck in standard conveyor belts

Width of belt, inches	Plies Minimum	Plies Maximum	Width of belt, inches	Plies Minimum	Plies Maximum
12	3	4	30	5	8
14	3	5	36	5	9
16	3	5	42	6	10
18	4	6	48	6	11
20	4	6	54	7	13
24	4	7	60	7	14

The speed at which a conveyor belt travels should depend upon the load it carries. Generally, it should be as slow as possible consistent with uniform maximum loading. If a belt is underloaded the load will be carried in the middle, and the center will wear out faster. If the load carried by a belt already installed is less than its capacity at the operating speed used the speed should be reduced. Recommended maximum speeds are given in table 64.

TABLE 64.- Maximum recommended speeds, feet per minute

Belt width, inches	Speed of general-purpose belts	Speed of belts for abrasive materials
12	300	250
14	300	250
16	300	250
18	350 - 400	300
20	350 - 400	300
24	400 - 500	350
30	450 - 500	350
36 - 42 - 48	500 - 600	400
54 - 60	600	400

Conveyor belts of the same specifications made by various manufacturers will vary somewhat in unit weight, but the difference will be small. The average unit weight per ply, exclusive of rubber covers, may be calculated as follows.

	Weight per ply per inch of width per linear foot			
Duck	28 oz.	32 oz.	36 oz.	42 oz.
Pounds	0.021	0.024	0.027	0.032

The rubber cover on the load-carrying side of the belt ranges in thickness by sixteenths from 1/16 to 1/4 inch. That on the pulley side is usually not over 1/32 inch thick but may be 1/16 or even 1/8 inch for severe service. The weight of the rubber covering may be calculated as 0.018 pound per 1/32 inch of thickness per inch of width per linear foot.

A 5-ply, 30-inch conveyor belt of 28-ounce duck having a 1/8-inch load cover and a 1/32-inch pulley cover would then weigh approximately

		Pounds per foot
5 x 30 x .021	=	3.15
5 x 30 x .018	=	2.70
		5.85

Service Equipment Required

The only way in which the load carried can cause wear on a conveyor belt is through abrasion, and abrasion requires motion. If an object carried by a conveyor does not change its position with respect to the belt while in transit it causes no wear. The field conveyor should be designed to carry its load with the least change of load position while in transit. For this reason the method of loading a conveyor belt has a direct bearing upon the life of the belt. Field conveyors should never be loaded directly by mechanical excavators. When they are so loaded the gravel is moving vertically downward when it strikes the belt. The belt must convert this downward motion to one coinciding with its own line of travel. This requires time, during which the gravel is changing its position on the belt and causing abrasion. The motion due to impact also causes abrasion. In addition, mechanical excavators deliver material in batches which, if loaded directly, would cause concentrated loads at irregular intervals on the belt, resulting in a constant shifting of position of the load as it passed over each idler and excessive stresses in the belt. To obviate intermittent loading a portable hopper is usually set up over the belt into which the excavator drops its load. It may be simply a wooden or steel box with sloping sides, terminating in a chute over the conveyor. Such a hopper is usually an independent unit and is picked up and moved ahead by the excavator. The hopper may be part and built integrally with the end section of the conveyor. The type used ordinarily depends upon local conditions. Regardless of type, the design should be such as will convert the batch loading of the excavator to a regular and continuous feed to the belt.

The feed to the belt although continuous may be irregular if uncontrolled, hence the feed from the hopper must be controlled at all times. If the gravel is dry, free from lumps of clay, and graded in size so as to permit it to run freely then the simplest type of control is a chute closed by a sliding gate operated by rack and pinion. The gate can then be set at an opening that will permit the passage of gravel at the required rate to the belt.

If the gravel is not free-running, owing to contained moisture, clay, or boulders, the gate-controlled chute is not a very satisfactory feeder because of the tendency of the material to stick in the opening. If the narrowest part of the chute can be made three times the size of the average boulders little trouble should be experienced from choking. With sticky clay the size of the opening makes little difference, as the clay will stick and cause trouble anyway. Often, a small stream of water played on the chute facilitates movement of sticky gravel.

If the gravel is free from clay but contains considerable coarse material or boulders it is a good plan to construct the bottom of the chute of wire screen or grizzly bars. This permits the sand and fine gravel to drop through ahead of the coarse material and form a cushion on the belt, thus reducing the impact when boulders are received. Screens or grizzlies are not very satisfactory if the gravel contains clay because of its tendency to stop the openings.

Where chutes and gates do not afford good feed control some form of mechanical feeder should be installed. There are several types available, among which are the apron feeder, the reciprocating feeder, and the rotary or barrel feeder. The apron feeder (fig. 20) may be a short length of conveyor belt taking material from the chute and delivering it to the conveyor, or it may be built of steel with interlocking pans. The reciprocating feeder (fig. 21) is driven by one or more eccentrics. The carrying floor of the feeder may be horizontal or on an angle, as shown. It may be built of a single flat plate or screen, or it may be made of grizzly bars, each of which is set differently on the eccentric. If a screen or grizzly is used the conveyor extends back, as shown by dotted lines, and the fines fall through the feeder to form a cushion for the boulders. The roll or barrel feeder (fig. 22) is simply a cast-iron or steel cylinder revolving below the chute opening. All feeders obtain their material from an adjustable opening in the hopper bottom. Skirt boards should always be provided to prevent spillage as the gravel falls to the belt.

Conveyor idlers should never be placed where the falling material strikes the belt. By directing the loading to a point between rolls the impact is absorbed by the flexibility of the belt, and less abrasion occurs.

Irrespective of the type of feeder used the feed should be directed so that it moves in the same direction as the belt and at about the same speed. This is not always possible, but the closer the delivery approaches these ideal conditions the less abrasion there will be due to impact and subsequent movement on the belt.

Capacity

The capacity of a field conveyor depends upon the method of loading, the cross-sectional area of the load, the width and speed of the belt, and the weight per cubic foot of the material handled. If the loading method is inadequate to furnish the maximum cross-sectional area the belt can carry, or does not supply a uniform flow of material to the belt, the capacity is

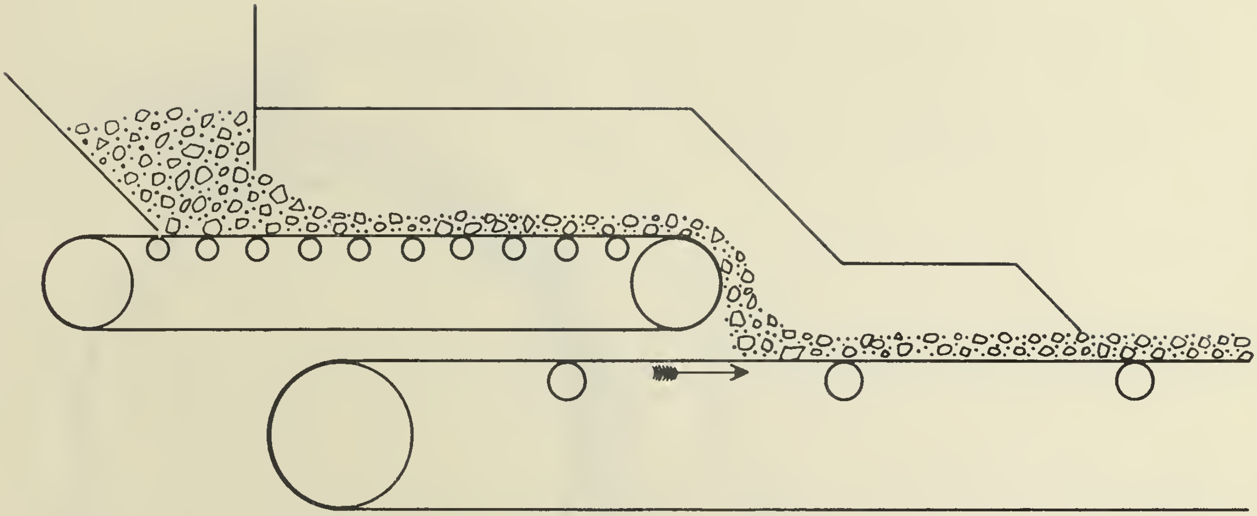


Figure 20.— Apron feeder.

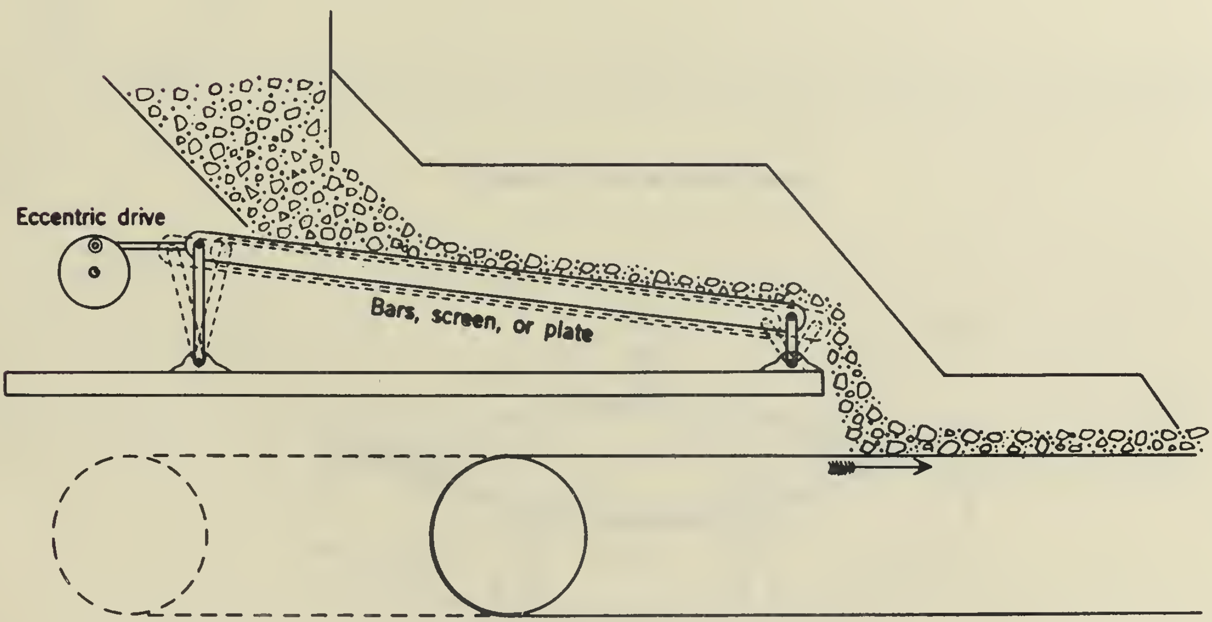


Figure 21.— Reciprocating feeder.

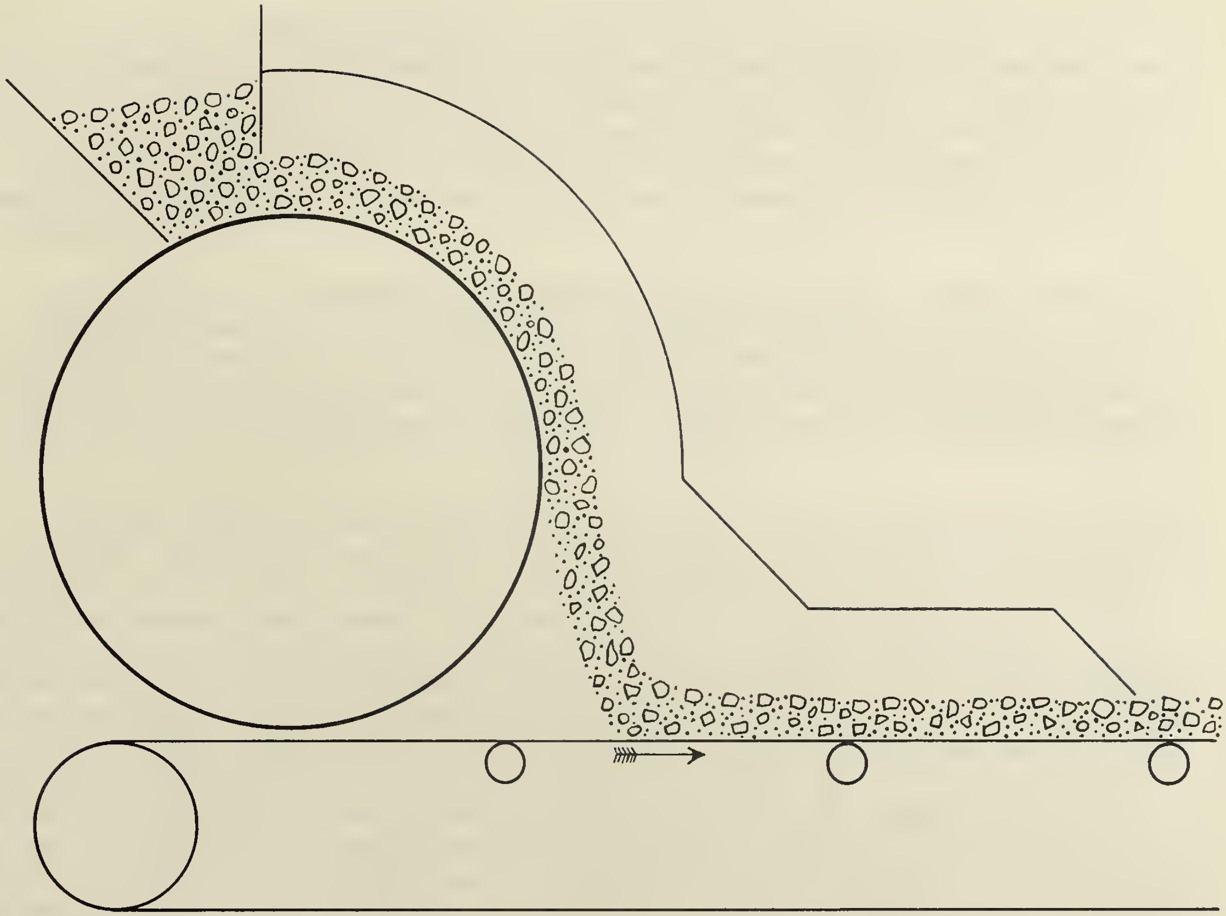


Figure 22.— Roll or barrel feeder.

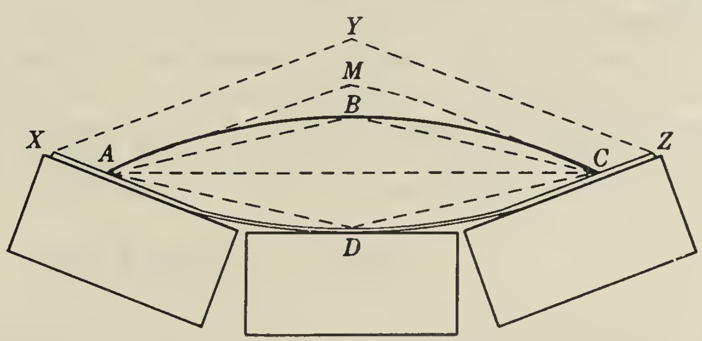


Figure 23.— Cross-sectional area of load.

correspondingly reduced. Therefore, the method of loading requires as careful consideration in calculating capacity as in calculating its effect upon the life of the belt. If the loading device cannot supply the maximum load which the belt width and speed allow, then the speed of the conveyor should be reduced. In the following discussion of conveyor capacities it is assumed that the design of the loading device is adequate for maximum belt capacity.

The trough or cross-sectional shape of the belt has a direct bearing on its haulage capacity. Flat belts have only about half the capacity of properly troughed belts. The proper trough shape depends upon the width and construction of the belt, the type of material carried, and the size, shape, and gradation of the individual particles. Hence, proper troughing depends upon the factors involved in each belt installation.

A field conveyor handling bank-run sand and gravel carries most of the load on the center of the belt. Theoretically, a belt can carry a load the cross-section of which coincides with the shape of the belt on the bottom and slopes upward and inward from each edge on an angle equivalent to the angle of repose of the gravel. Practically, the load never extends to the extreme edge of the belt, except when it is overloaded, and the upper part of the cross-section has a rounded perimeter due to settling as the belt passes over the idlers. The maximum cross-sectional area will thus depend upon the physical characteristics of the load itself, as well as upon the shape of the trough. For this reason the capacity of a belt may change from day to day as the physical characteristics of the gravel change in the working face of the pit. Wet weather may increase the moisture content of the gravel and cause it to build up higher on the belt. If the gravel is too wet this effect may be reversed, the load slumping to the edges. A change in the ratio of coarse to fine material in the feed is also reflected in the shape of the load.

Figure 23 shows the cross-section of a loaded conveyor belt. The area XYZD represents the theoretical load, impossible to reach because of settling and spillage. Area AMCD represents the section when first placed on the belt; this settles to area ABCD. The lateral angles will of course vary for different classes of material. A common method of calculating the average cross-sectional area of the load is to let the quadrangle ABCD (dotted lines) represent that area. The length AC is assumed as 0.8 the width of the belt and BD as 0.2 the width. If W is the width of the belt, the area is

$$\frac{0.8W \times 0.2W}{2} = 0.08W^2.$$

The maximum capacity, in cubic feet of material, may then be computed by the following formula.

$$L = \frac{0.08W^2 \times S \times 60}{144}$$

in which

L = the cubic feet of load per hour;
W = the width of the belt, in inches;
S = the speed of the belt, in feet per minute.

At a speed of 100 feet per minute the maximum capacity of the conveyor is then

$$L = 3\ 1/3\ W^2 \text{ cubic feet per hour.}$$

This formula is predicated upon the assumption that the cross-sectional area of the load varies directly with the square of the width of the belt. However, experiments have shown that with the same type of material the load can be spread relatively closer to the edges on a wide than on a narrow belt. Therefore, the square of the belt width must be multiplied by a factor which varies with the width of the belt. This factor has been shown empirically to range from 3.2 for a 12-inch belt to 4.0 for a 60-inch belt, assuming a belt speed of 100 feet per minute.

If this factor is designated as K, table 65 gives its value for belts from 12 to 60 inches wide in increments of 2 inches.

TABLE 65.- Values of K for various belt widths

Belt width, inches	K	Belt width, inches	K	Belt width, inches	K	Belt width, inches	K	Belt width, inches	K
12	3.2	22	3.37	32	3.53	42	3.70	52	3.87
14	3.23	24	3.40	34	3.57	44	3.73	54	3.90
16	3.27	26	3.43	36	3.60	46	3.77	56	3.93
18	3.30	28	3.47	38	3.63	48	3.80	58	3.97
20	3.33	30	3.50	40	3.67	50	3.83	60	4.00

The maximum belt capacity may be obtained from the following formula,

$$C = \frac{K \times W^2 \times S \times M}{200{,}000},$$

in which

C = the capacity, in tons per hour;
K = the factor of load cross-section;
W = the width of the belt, in inches;
S = the speed of the belt, in feet per minute;
M = the weight of the load per cubic foot, in pounds

Sand and gravel ranges in weight from 2,900 to 3,240 pounds per cubic yard, (107.4 to 120 pounds per cubic foot). Table 66 gives the range in maximum capacity, in tons per hour, for belts of various widths operating at a speed of 100 feet per minute and carrying materials of different weights per cubic foot. Table 66, A, gives the maximum capacity of belts of various widths handling sand and gravel at various speeds.

TABLE 66.- Maximum conveyor-belt capacities, in tons per hour at belt speed of 100 feet per minute

Belt width, inches	Factor K	Weight of material, pounds per cubic foot				
		100	107.4	110	115	120
12	3.2	23.0	24.7	25.3	26.5	27.6
14	3.23	31.6	34.0	34.8	36.4	38.0
16	3.27	41.9	45.0	46.0	48.1	50.2
18	3.30	53.4	57.4	58.8	61.5	64.2
20	3.33	66.6	71.5	73.3	76.6	80.0
24	3.40	98.0	105	108	112	118
30	3.50	157	169	173	181	189
36	3.60	233	251	257	268	280
42	3.70	327	350	359	375	392
48	3.80	438	470	482	504	525
54	3.90	568	610	626	654	682
60	4.00	720	774	792	828	864

TABLE 66, A. - Maximum average capacities of belt conveyors, tons per hour[1]
(Sand and Gravel)

Belt width, inches	Belt speed, feet per minute (gravel ranging from 107.4 to 120 pounds per cubic foot)											
	100		150		200		250		300		350	
	From	To	From	To	From	To	From	To	From	To	From	To
12	24.7	27.6	37.1	41.5	49.5	55.3	61.9	69.2	74.3	83.0		
14	34.0	38.0	51.0	57.0	68.0	76.0	85.0	95.0	102	114		
16	45.0	50.2	67.4	75.3	90.0	100	112	125	135	151		
18	57.4	64.2	86.1	96.2	115	128	143	160	172	192	201	224
20	71.5	80	107	120	143	160	179	200	215	240	250	280
24	105	118	158	176	210	235	263	294	316	352	368	411
30	169	189	254	283	338	378	423	472	507	567	592	662
36	251	280	376	420	500	560	627	700	752	840	878	980
42	350	392	526	588	702	784	877	980	1,050	1,175	1,225	1,370
48	470	525	706	788	940	1,050	1,175	1,310	1,410	1,575	1,645	1,840
54	610	682	916	1,020	1,220	1,360	1,525	1,700	1,830	2,050	2,140	2,390
60	774	864	1,160	1,300	1,550	1,730	1,930	2,160	2,320	2,590	2,710	3,025

Belt width, inches	Belt speed, feet per minute (gravel ranging from 107.4 to 120 pounds per cubic foot)									
	400		450		500		550		600	
	From	To	From	To	From	To	From	To	From	To
12										
14										
16										
18	230	257								
20	286	320								
24	420	470	473	529	526	588				
30	676	756	762	851	846	945				
36	1,000	1,120	1,127	1,260	1,250	1,400	1,380	1,540	1,500	1,680
42	1,390	1,565	1,575	1,760	1,750	1,960	1,920	2,150	2,100	2,350
48	1,880	2,100	2,120	2,360	2,350	2,630	2,590	2,890	2,820	3,150
54	2,440	2,730	2,750	3,070	3,050	3,410	3,360	3,750	3,660	4,100
60	3,090	3,460	3,480	3,890	3,860	4,320	4,250	4,750	4,640	5,180

1/ Working capacities probably do not exceed 60 to 75 percent of these quantities.

The capacities shown in the tables are based on the following assumptions: (1) The loading facilities are adequate to deliver the load to the belt at its maximum carrying rate; (2) the size gradation of the gravel is suitable for the width, speed, and inclination of the belt; (3) the applied power is ample; (4) the conveyor construction is such as to give the proper trough shape for the gravel handled; and (5) the belt has sufficient strength to withstand the stresses set up.

It must also be remembered that the capacities in table 66 are the average maximum for a belt speed of 100 feet per minute. For belts running at greater speeds the figures should be multiplied by the speed in hundreds of feet per minute.

The average working capacities will always be less than the tabulated figures because of imperfect loading and delays.

If the gravel contains a large percentage of boulders, their size fixes the width of the belt (see table 59), and the speed should be as slow as possible to give the required capacity.

With a high ratio of sand and fine gravel the capacity fixes the belt width, but here again the speed should be as slow as possible.

A slow-speed belt will carry a deeper load than a high-speed one.

Average working capacities probably will not exceed 75 percent of the calculated maximum rate.

Power

The power used to operate a conveyor belt may be taken from any type of prime mover, such as a steam or compressed-air engine; gasoline, Diesel, or electric motor; or a line shaft. Field conveyors, as commonly installed, are operated through speed reducers from electric motors. Speed may be reduced through belts and pulleys, gears, rope drives, or chain and sprockets. Enclosed geared speed reducers are usually employed on field conveyors.

The power may be applied to the belt at the head pulley, at the tail pulley, or at any convenient intermediate point. Field conveyors are commonly driven from the discharge or head pulley and less often from the tail pulley, and where high belt tension is necessary the drive is usually independent of the conveyor terminal pulleys.

The power required to drive a belt conveyor depends upon the total weight moved, the frictional resistance to movement, and the height through which the pay load is elevated.

The total weight moved is the sum of the weights of the pay load, the belt itself, the moving parts of the supporting rolls, and all terminal, tension, and drive pulleys. The weight of the pay load depends upon its average

cross-sectional area, its weight per cubic foot, and the length of the conveyor. The weight of the belt depends upon its width, length, the number of plies of duck, and the thickness of its rubber cover. The weight of the various pulleys depends on their size and the material of which they are built. The weight of the supporting rolls depends on the material of which they are constructed (steel tubing or cast iron) and their spacing along the conveyor.

The frictional resistance depends upon the type of bearings used in the several pulleys and rolls. This ranges from a simple shaft and pillow block through various types of roller or ball bearings to the latest type of improved antifriction construction. The frictional resistance depends also upon the type of lubricant used and the efficiency of application. Accurate calculation, based on a careful analysis of weight and friction coefficients, may be entirely erroneous in practical application if the lubrication is faulty.

The calculation of power required to elevate the pay load involves merely the weight lifted and the elevation. Sometimes calculation of this part of the power required may involve a percentage of the weight of the belt on the inclined portion of the conveyor, but the effect is so small that it is usually neglected.

If the conveyor operates on a descending rather than an ascending grade, the power developed by the load moving down-grade must be deducted from that required to move the load and overcome friction. If the power so generated is more than that required to move the load and overcome friction, some braking or hold-back device must be applied to control the belt speed.

In calculating the power required to move the belt it is customary to assume that the belt is uniformly loaded to its maximum average capacity. While it is true that in practical operation few belts are so loaded continuously the installed power must be sufficient to provide for such conditions.

From the preceding discussion it is evident that an accurate computation of power requirements can be made only when all the facts are known, and then only as they affect each installation. Conveyor-belt manufacturers use various formulas for computation of the required power. If a given set of operating conditions is assumed the power requirements calculated from the formulas may vary as much as 10 percent, probably owing to the variation in the values assigned to the average weight of the moving parts, the coefficient of friction, and the weight of the belt itself.

Calculation of power requirements by formulas must then be considered approximate rather than exact, unless all factors are accurately known. For this reason the author prefers to establish the minimum and maximum limits for variable factors, from which can be determined the minimum and maximum limits of the required power.

The range in weight of the moving parts of belt conveyors, calculated per foot of conveyor length and including 2 feet of belt, has been abstracted from manufacturers' catalogs and is shown in table 67.

TABLE 67.- Weight of the Moving parts of conveyor belts, including 2 feet of belt

Belt width, inches	Weight per foot of conveyor, pounds	
	Minimum	Maximum
12	12	15
14	15	17
16	17	20
18	19	24
20	21	28
24	25	35
30	32	47
36	40	63
42	49	76
48	59	91
54	69	107
60	81	125

The factors of frictional resistance, as determined empirically for conveyor belts, are as follows:

For plain greased bearings - - - - - - - - - - - 0.06
For average antifriction bearings - - - - - - - - .04
For improved antifriction bearings - - - - - - - .025

Table 68 shows the weight of the average maximum load per foot of conveyor, computed from the capacities given in table 66.

TABLE 68.- Weight of load per foot of conveyor, pounds

Belt width, inches	Weight of material per cubic foot, pounds				
	100	107.4	110	115	120
12	7.7	8.2	8.4	8.8	9.2
14	10.5	11.3	11.6	12.1	12.7
16	14.0	15.0	15.3	16.0	16.7
18	17.8	19.1	19.6	20.5	21.4
20	22.2	23.8	24.4	25.5	27.0
24	32.7	35.0	36.0	37.3	39.2
30	52.3	56.3	57.7	60.3	63.0
36	77.7	83.7	85.7	89.3	93.3
42	109.0	115.7	119.7	125.0	131.0
48	146.0	157.0	160.7	168.0	175.0
54	189.0	203.3	208.7	218.0	227.0
60	240.0	258.0	264.0	276.0	288.0

The power required depends directly upon the force or pull necessary to move the belt and its load. This pull may be divided into three components:

(1.) That necessary to move the empty belt horizontally;
(2.) that necessary to move the load horizontally; and
(3.) that necessary to move the load vertically.

For extreme accuracy a fourth component should be considered, namely, that necessary to raise the empty belt on the inclined portion of the conveyor, but this is small and is customarily omitted.

The horizontal pull required is computed by multiplying the total weight moved by the friction factor as follows:

$$hP = W \times L \times F,$$

in which

hP = horizontal pull, in pounds;
W = total weight per foot of conveyor, in pounds;
L = length of the conveyor, in feet;
F = factor of frictional resistance.

Then

$$W = w + m$$

in which

w = the weight of load per foot of conveyor (table 68), and
m = the weight of moving parts per foot of conveyor (table 67).

The vertical pull required by the load on 1 foot of the conveyor is found by multiplying its weight by the vertical distance through which it moves. Thus

$$vP = wH,$$

in which

vP = the vertical pull required to move the load carried on 1 foot of the conveyor, pounds; and
H = the vertical distance the load is elevated, feet,

The total pull required to operate the conveyor depends upon whether the conveyor is level, elevates the load, or lowers the load.

Total pull $P = hP$ for horizontal conveyors.

Total pull $P = hP + vP$ for elevating conveyors.

Total pull $P = hP - vP$ for lowering conveyors.

The power required may be found from the formula

$$HP = \frac{P \times S}{33{,}000},$$

in which

HP = theoretical horsepower;
P = total pull required, pounds;
S = speed of the conveyor, feet per minute.

Example 11

What horsepower would be required to operate a belt conveyor 36 inches wide and 500 feet long, using average antifriction bearings, running 300 feet per minute, and carrying sand and gravel weighing 110 pounds per cubic foot up an incline with a total elevation of 50 feet?

From table 67 the weight (m) of the moving parts may range from 40 to 63 pounds per foot. The weight is assumed as 50 pounds.

From table 68 the weight (w) of the load is 85.7 pounds per foot.
Then

$$W = 85.7 + 50 = 135.7 \text{ pounds.}$$

Moreover,

		Pounds
hP = W x L x F	= 135.7 x 500 x 0.04	= 2,714
vP = wH	= 85.7 x 50	= 4,285
P =		6,999

Then

$$HP = \frac{6,999 \times 300}{33,000} = 63.6$$

If the conveyor had been level the power required would be

$$\frac{2,714 \times 300}{33,000} = 24.7 \text{ HP}$$

If the conveyor lowered the gravel 50 feet the power required would be negative, or the power developed would be

$$\frac{(4,285 - 2,714) \times 300}{33,000} = 14.3 \text{ HP.}$$

These calculations do not include the power required to overcome friction in the driving mechanism. For enclosed-geared speed reducers and for each reduction by cut gears or belt and pulley 5 percent should be added for horizontal or elevating conveyors and subtracted for lowering conveyors. In addition, 10 percent should be added for surplus power to take care of temporary overload in determining the brake horsepower of the motor for the horizontal or elevating conveyor and subtracted for the lowering conveyor.

Belt Ply

The pull necessary to move the belt and its load horizontally, plus the pull necessary to elevate the load, constitute the "effective" pull or tension required to operate a belt conveyor. However, they do not equal the total tension that the belt must withstand. A conveyor belt may be visualized as reacting the same as any belt used to transmit power. If the belt is merely laid over the drive pulley without tightening, the pulley will revolve and slip beneath the belt without moving it. If the two ends of the belt are fastened together around another pulley and the belt is tightened, the belt

will revolve with the drive pulley due to the friction between the belt and the face of the pulley. As the tension on the belt is increased the pulley friction increases and the belt's capacity to transmit power also increases. This tension, called "initial tension", is necessary before motion takes place in the belt and is independent of the tension necessary to move or elevate the load. However, increasing pulley friction by belt tension alone may set up undue stresses in the belt itself, and the tension must be limited by the strength of the belt which depends on the number of plies of duck it contains.

Pulley friction may be increased in two ways without increasing belt tension: (1) By increasing the arc of contact between drive pulley and belt and (2) by lagging the face of the drive pulley with rubber or some other material of a greater friction coefficient with the belt than the bare pulley face. Thus, an increase in pulley friction by either or both methods will permit application of more power without any increase in belt tension. This is an important factor in belt-conveyor design, since the construction of the belt and its ability to transmit power will depend upon the type of drive used.

Single, unlagged pulleys used without snubbers afford an arc of contact of only 180° and a minimum of friction. By using snubber pulleys the arc may be increased to 240°. Double-pulley or tandem drive affords an arc of contact of 360°, which may be increased to 480° or more by the use of snubbers. The friction in both types of drive may be increased by lagging the pulley faces.

The use of both tandem drives and snubber pulleys, however, flexes the belt in opposite directions, and the belt wraps around the second pulley with its load-carrying side in contact. This increases internal stress due to flexure and provides a chance for excessive abrasion from fine particles clinging to the belt and being ground against the pulley face. Abrasion is reduced if the pulleys are lagged.

Other types of drive have been developed for conveyor belts but those mentioned are in most common use.

The total tension to which a conveyor belt is subjected is then

$$T = P + S,$$

in which

T = total belt tension, pounds;
P = total belt pull or effective tension, pounds;
S = tightening or slack-side tension.

The value of S, or the slack-side tension in the belt, is produced by a counterweighted take-up device or by a screw take-up and is usually represented by a percentage of the effective tension, P; it depends upon the arc of contact with the drive pulleys and the coefficient of friction for bare or lagged pulleys.

With a screw take-up for applying tension there is no way of measuring accurately the tension applied, whereas with a counterweighted take-up the tension can be controlled by the weight.

Table 69 gives the average slack-side tension for counterweighted take-up drives in percentage of effective tension or belt pull.

TABLE 69.- Average slack-side tension in percentage of effective tension

Arc of contact, degrees	Counterweighted take-up	
	Bare pulleys, percent	Lagged pulleys, percent
180	100	52
210	70	40
240	54	32
360	26	13
420	20	9
480	16	6

From the values given in table 69 for slack-side tension, S, and the working stress of belt plies in table 61, the necessary number of plies and weight of duck can be computed by the following formula.

$$PD = \frac{T}{WX},$$

in which

PD = number of plies of duck;

T = total belt tension, in pounds;

W = width of belt, in inches;

X = ply-inch working stress of belt, in pounds.

Example 12

In example 11 the effective tension or belt pull was calculated to be 6,999 pounds. With application of a single, bare, unsnubbed head-pulley drive the total belt tension would be

$$T = P + S = 6,999 + (1.00 \times 6,999) = 13,998 \text{ pounds.}$$

Using 28-ounce duck with a maximum ply-inch stress of 25 pounds, the number of plies required would be

$$\frac{13,998}{36 \times 25} = 15.5$$

From table 58 the maximum number of plies permissible for proper troughing with a 36-inch belt was 9, hence some other type of drive will be required.

If a tandem drive is assumed with an arc of 420°, obtained by snubbing and lagged pulleys, the total tension will be

$$T = P + S = 6,999 + (0.09 \times 6,999) = 7,629 \text{ pounds.}$$

With 28-ounce duck at 25 pounds ply-inch stress the number of plies required would be

$$\frac{7,629}{36 \times 25} = 9.$$

PUMPS AND PIPE LINES

Centrifugal pumps are used at many plants to convey sand and gravel through pipe lines. The pump and its discharge line may constitute the sole method of transportation, as in a dredge pump delivering direct to the treatment plant. It may be supplementary to other means of transportation, as in trucks hauling from a dry pit and dumping to a sump from which the pump delivers the gravel to the plant. Another instance is conveyance from pit to sump by sluices supplemented by pumps from sump to plant. Conversely the pump and pipe line may comprise the primary means of transportation, supplemented by other types as in a dredge pump delivering to a dewatering bin on shore from which a conveyor belt takes the gravel to the plant.

Pump and pipe line transportation may be designed so that two or more pumps operate in series in a single pipe line. The dredge pump in this case is supplemented by one or more booster pumps in the discharge line. Under different conditions the dredge pump may discharge to sump or underwater storage and a second pump take material from there to the plant, thus cutting the haulage service into two or more independent units. A dragline, power scraper, or slackline cableway may be substituted for the last pump. Obviously, the centrifugal pump may be used in a number of ways.

The selection of pumps for transportation depends upon a number of factors such as: (1) The quantity of material to be transported, (2) the presence of enough water as a transporting medium, (3) the length of the haul, (4) the height to which the material must be elevated, (5) the physical character of the gravel, (6) the power required, and (7) the cost of installation.

The quantity of material to be transported obviously governs the size of pump and pipe line. The movement of quantities in excess of a certain limit - determined by local conditions - results in excessive expense for maintenance and power over that required for other haulage equipment, such as barges.

TABLE - 70. - Composition of sand, gravel, and water mixtures

(Sand and gravel assumed to weigh 110 pounds per cubic foot and contain 40 percent voids

Sand and gravel by volume, percent	100 cubic feet of mixture will contain -					1 cubic yard of sand[1] and gravel will require water as follows -			Ratio of sand and gravel to water			
	Sand and gravel		Water						By volume		By weight	
	Cubic feet	Pounds	Cubic feet	Pounds	U. S. gallons	Cubic feet	Pounds	U. S. gallons	Sand and gravel	Water	Sand and gravel	Water
5	5	550	97.0	6,060	726	524	32,750	3,920	1	19.4	1	11.0
6	6	660	96.4	6,020	721	434	27,120	3,245	1	16.0	1	9.1
7	7	770	95.8	5,990	716	370	23,120	2,770	1	13.7	1	7.8
8	8	880	95.2	5,950	712	321	20,070	2,400	1	11.9	1	6.8
9	9	990	94.6	5,910	708	284	17,750	2,125	1	10.5	1	6.0
10	10	1,100	94.0	5,875	703	254	15,875	1,900	1	9.4	1	5.3
11	11	1,210	93.4	5,840	698	230	14,375	1,720	1	8.5	1	4.8
12	12	1,320	92.8	5,800	694	209	13,050	1,563	1	7.7	1	4.4
13	13	1,430	92.2	5,760	690	192	12,000	1,435	1	7.1	1	4.0
14	14	1,540	91.6	5,725	685	177	11,050	1,324	1	6.5	1	3.7
15	15	1,650	91.0	5,690	681	164	10,250	1,226	1	6.1	1	3.45
20	20	2,200	88.0	5,500	658	119	7,440	890	1	4.4	1	2.5
25	25	2,750	85.0	5,310	636	92	5,750	688	1	3.4	1	1.9
30	30	3,300	82.0	5,120	613	74	4,625	554	1	2.7	1	1.55
40	40	4,400	76.0	4,750	568	51	3,190	381	1	1.9	1	1.08

1/ 1 ton of sand and gravel will require approximately two thirds the tabulated figures.

In handling sand and gravel through pipe lines the transporting medium is water. Under average operating conditions the sand and gravel conveyed constitute 10 to 15 percent by volume of the mixture pumped, the balance being water. Sand and gravel contain voids or unfilled spaces between particles. The percentage of voids varies with the size gradation of the gravel and sand. When the mixture is moved by pumping these voids are filled with water, hence the actual quantity of water present in a given volume is always greater than the complementary percentage of sand and gravel. If sand and gravel are assumed to average 110 pounds per cubic foot loose measure and to contain 40 percent voids the relative quantities of water and gravel for various mixtures is as given in table 70. After the percentage of solids is once determined the water required as a vehicle can be read from the table. Correction should of course be made for variation in weight and void content of the gravel. When gravel is dredged from a running stream there is assurance of enough vehicular water. When it is dredged from an artificial pond in which the subsurface water movement is retarded additional water may be necessary from an independent source. Again, pumps handling material from a sump may require more water than enters the sump with the gravel, although ordinarily the feed to the pump can be made more uniform, resulting in a heavier percentage of solids and requiring less water than in original pumping or sluicing.

The head against which the material must be pumped is affected by the length of the pipe line and the height to which the material must be elevated. The length and diameter of the pipe determine the friction head, and the elevation determines the static head.

The physical characteristics of the sand and gravel will affect the pumping operation in various ways, all of which, while evident, are difficult to evaluate accurately. Round particles create less internal pipe friction than flat or sharp-edged particles; coarse gravel creates more friction than sand and requires a higher velocity of flow; and increased percentages of solids, either coarse or fine, increase friction and require greater velocity. The direct effect of these characteristics must be determined by trial or from the past experience of the operator.

The power required is determined by the quantity pumped, the percentage of solids, the total pumping head, and the operating characteristics of the pump. Power is probably the factor of greatest economic importance in the selection of a pump and pipe lines as a means of transportation, as is illustrated in the following example:

<u>Example 13</u>

It is assumed an operator is pumping a mixture containing 12 percent solids against a total head of 25 feet at the rate of 100 tons of sand and gravel per hour and wishes to increase the head to 100 feet and put the material directly to the top of his washing plant.

If the sand and gravel weigh 110 pounds per cubic foot the required quantity (100 tons per hour) in cubic feet per minute will be

$$\frac{100 \times 2000}{60 \times 110} \text{ or } 30.3 \text{ cubic feet per minute.}$$

From table 70, 30.3 cubic feet of sand and gravel at 12 percent solids will require

$$12 : 92.8 :: 30.3 : X = 234 \text{ cubic feet of water.}$$

Assuming 40 percent voids and allowing for the water filling the voids the volume of mixture pumped per minute will be

$$30.3 - \frac{40 \times 30.3}{100} + 234 = 252 \text{ cubic feet.}$$

The weight of the mixture per cubic foot will be

	pounds
30.3 cubic feet of gravel at 110 pounds	3,333
234 cubic feet of water at 62.5 pounds	14,625
Total	17,958

Then

$$\frac{17,958}{252} = 71.3 \text{ pounds per cubic foot.}$$

The theoretical or water horsepower required can be computed from the formula

$$HP = \frac{Q\ W\ H}{33,000},$$

in which HP = the theoretical or water horsepower,

Q = the quantity of mixture handled, in cubic feet per minute;
W = the weight of the mixture per cubic foot;
H = the total head, in feet.

If there is no increase in the length of the discharge pipe required by the increase in total head substitution in the formula gives the following power requirements:

$$HP = \frac{252 \times 71.3 \times 25}{33000} = 13.6, \text{ to pump the mixture against a 25-foot head.}$$

$$HP = \frac{252 \times 71.3 \times 100}{33000} = 54.4, \text{ to pump the mixture against a 100-foot head.}$$

The motor rating in each case should be about double these figures. The operator will then be faced with the problem of changing from a 30-to a 110-HP motor. He must evaluate this against the economics involved in dewatering his pump discharge and using some other type of elevator to gain the added 75 feet of elevation. Possibly a belt conveyor or a bucket elevator would do the work with less than the 80 horsepower involved. On the other hand, he may be transporting water to the top of his plant for washing purposes.

It is interesting to note that if the gravel could be transported dry the power required would be only about 20 percent of that necessary to convey the mixture of gravel and water.

$$\frac{30.3 \times 110 \times 25}{33000} = 2.5 \text{ HP to pump gravel only against 25-foot head.}$$

$$\frac{30.3 \times 110 \times 100}{33000} = 10.1 \text{ HP to pump gravel only against 100-foot head.}$$

In other words, 80 percent of the power is used to transport the water.

Besides these factors, others may influence the operator in choosing this type of transportation equipment. Many gravel deposits contain objectionable quantities of sticky clay, which must be removed to meet specifications for washed aggregates. The churning action given the material in its passage through pump and pipe line and the solvent effect of the water provide a very effective preliminary washing. In some instances no further wash water may be required. In others, further rinsing may be necessary, but less additional water will be needed than if the gravel had been delivered dry to the plant.

Pumps and pipe lines share with sluices the advantage of being the only types of transportation which perform the dual function of washing and transportion.

Thorough wetting of the sand and gravel particles before their delivery to the treatment plant facilitates treatment in the plant. Thoroughly wet material accompanied by a relatively large body of water requires less time in screening. The same plant fed by dry material must provide some method of wetting either on the initial screen or before the material reaches it.

On the other hand, pump and pipe-line transportation probably requires more water to be handled than would be needed for washing only. Welch[5/] estimates the average water requirement for washing in gravel plants as 1 gallon per minute per cubic yard per 10 hours of plant capacity but adds that more water is better. A study of a number of plants by the author reveals that the ratio of sand and gravel to water for washing ranges from 1 ton of sand and gravel to 1.2 to 4.1 tons of water.

5/ Welch, F. M., Design of Sand and Gravel Washing and Screening Plants: Rock Products, vol. 32, no. 15, Aug. 31, 1929, p. 52.

Applying Welch's average requirements to example 13 the water required would be 667 gallons per minute, and the ratio of sand and gravel to water by weight would be 1 to 1.7. Pumping with 12 percent of solids, the ratio of sand and gravel to water in the pipe discharge (from table 70) would be 1 to 4.4 by weight or 2-1/2 times the average quantity calculated. However, all the vehicular water cannot be used for washing, and additional water must be added for rinsing.

Pump and pipe-line transportation is advantageous for relatively short distances or average hydraulic heads. Procuring and delivering the gravel in one operation often eliminate intermediate storage and rehandling. The gravel is prewashed before it enters the washing plant, and a constant supply is provided to the treatment plant.

This method of transportation has the disadvantage of requiring a large proportion of additional power to transport the carrying water. It may be expensive due to the maintenance of floating pipe lines in river currents or in stormy weather.

Structural Limitations

Conveyance by pumps and pipe lines is limited to those gravel deposits which are comparatively free from oversize boulders. Ordinarily, with a deposit containing 10 percent of boulders larger than two thirds the diameter of the suction inlet of the pump this type of transportation could not be used.

No definite length of pipe discharge can be assigned for pipe-line transportation by a single pump or for one or more booster pumps in the line. The discharge length will depend upon the operating characteristics of the pumps used, the diameter of the pipe, and the character of the material pumped. However, certain limitations may be mentioned. Where booster pumps are necessary the material should enter the suction of the booster at some positive pressure. In other words, the booster should not be called upon to overcome any suction head. If the suction of the booster is put under a vacuum, leakage of air would probably result, causing pulsation in the flow. The amount of pressure required is only that necessary to insure against any vacuum in the booster suction at any time and is estimated by various engineers as ranging from 4.3 to 6.0 pounds per square inch, corresponding to 10 to 14 feet of head.

It is necessary also that the booster pump be so located that the discharge head or pressure is well within the bursting strength of pipe and pump. Sand and gravel traveling in long, straight, fairly level pipe lines tend to separate and drag along the bottom of the pipe. This may cause the line to fill up gradually and plug. The booster pump should therefore be placed as near the center of the discharge line as possible so that it may reagitate the material and prevent plugging. With a booster pump, if for any reason the dredge pump is stopped and there is no positive head on the suction of the booster, some means of relief must be provided or the line beyond the booster will also plug. Provision is commonly made for this by interlocking

the drives of both pumps or by inserting a bypass valve between the booster and the dredge pump. This valve is placed in a submerged depression in the discharge line with its emergency opening under water. Ordinarily the pressure in the pipe line holds this valve closed. When the pressure is removed by stoppage of the dredge pump or a plug in the line the valve opens and allows the booster pump to pull clear water into its suction and thus clear the line beyond. Such valves do not help if the stoppage occurs in the booster discharge line, hence interlocking drives are needed.

It has been shown that the power required to operate a pump and pipe-line transportation unit varies directly with: (1) The rate of discharge - governed by the diameter of the pipe and the velocity of flow, (2) the weight per cubic foot of the material pumped - governed by the percentage of solids, and (3) the total head against which the pump must operate - governed by the suction head, the friction head, and the static head. The head against which a pump can deliver depends upon the peripheral speed of its impeller, which in turn depends upon the diameter and revolutions per minute of the impeller or, in other words, the characteristics of the pump.

The total head against which a pipe-line conveyor operates may be constant or may vary from day to day. To illustrate, a pump obtaining material from a sump, supplied by other equipment, and transporting it to the treatment plant will operate under constant head. On the other hand, delivery from a constantly shifting dredge may or may not be subject to a corresponding change in head, depending on whether shifting the position of the dredge involves lengthening or shortening or no change in the discharge line. Consequently, a pump and pipe-line system subject to frequent change of head should be provided with means for varying the peripheral speed of the pump impeller. This is usually accomplished by variable-speed reduction between motor and pump. As an illustration of the seriousness of the effect of change of head Hawgood[6/] found that a pump handling sand and mud and operating at highest efficiency through a 4,000-foot discharge line required 10 HP per 100 feet of discharge line. When the discharge line was cut to 250 feet but the revolutions per minute of the pump were left as before, the power required rose to 108 HP per 100 feet of pipe. In other words, the efficiency of the pump varies inversely with the ratio of the peripheral speed of the impeller to the velocity equivalent of the total head, and highest efficiency results when the impeller speed approaches the velocity equivalent of the total head.

For a detailed discussion of the effect of design on pump efficiency the reader is referred to an article entitled "Economic Design of Hydraulic Pipe-Line Dredge," by John F. Cushman, published in Proceedings of the American Society of Civil Engineers, vol. 56, November 1930.

6/ Hawgood, H., The Hydraulic Transmission of Dredged Material at San Pedro Harbor, Calif.: Eng. News, vol. 62, no. 10, Sept. 2, 1909, pp. 244-6.

Auxiliary Equipment

Pump and pipe-line systems require a source of power. This source may be a steam engine or turbine or gasoline, Diesel, semi-Diesel, or electric motor. Combinations of Diesel, semi-Diesel or steam turbines with electric generators and motors may also be used.

The necessary priming apparatus is required, the characteristics of which depend upon the design of the individual pump.

While the pipe line is auxiliary to the pump, it is part of the system. Pipe lines, however, may be installed in various ways. They may be laid on the ground surface, floated on a water surface, or submerged and laid on a lake or river bed. In any case they must be flexible to provide facility in movement from place to place and to prevent breakage which is apt to occur from surges in the material pumped. On the other hand, adequate but flexible anchorage should be provided to prevent creeping of submerged or land lines.

Flexibility in the line is provided by rubber sleeves or balljoint connections between pipe sections at regular intervals. Excessive movement is prevented by fastening to "dead men" on the surface or anchors in the river bed. Floating lines are snubbed to anchored buoys.

Rubber-hose joints offer greater resistance or internal friction than ball joints but are less expensive in first cost. They are more liable to failure than ball joints but are usually replaced in less time when failure occurs. They are seldom used in modern practice on pipe larger than 15 inches in diameter.

Flexible joints on floating pipe lines should be placed between pontoon supports and not on the pontoon itself.

Pontoons are placed at various intervals in the line, depending on local conditions. Pipe is usually furnished in 20- to 40-foot lengths with flanged joints. Pontoons may be 30 to 60 feet apart but usually at not more than 40-foot centers.

The discharge pipe may be carried either parallel to the long axis of the pontoon or transverse. Pontoons of wooden or steel cylinders are usually built with two cylinders connected by a wooden or steel frame supporting the pipe cradle.

There is no uniformity in pontoon design, but from observed installations the ratio of the weight of the water which the pontoon is capable of displacing to the total weight supported ranges from 1.4 to 1 to 3.0 to 1. The pontoon must support its own weight, plus that of the pipe, plus that of the gravel and water in the pipe, hence if the weight of these three items is 1,000 pounds the pontoon should displace from 1,400 to 3,000 pounds of water when completely submerged.

Reference to the variation in weight between standard and spiral pipe, as given in table 72, will indicate that a pontoon suitable for spiral pipe may not be large enough for standard pipe.

Discharge lines offer the least frictional resistance when straight and when the ends at the joints are brought together without leaving gaps within the coupling or flange. All elbows and bends should be on as large a radius as possible.

The frictional resistance of various pipe-line accessories, expressed in equivalent feet of straight pipe, is given in table 71. For pipe friction and capacity and for head developed by dredge pumps the reader is referred to tables 32 to 38 in part 3 of this series of papers. Table 72 gives the principal characteristics of pipe commonly used in dredging service.

TABLE 71. - Friction equivalent (in feet of straight pipe) of loss in fittings

Fitting	Diameter of pipe, inches											
	4	5	6	8	10	12	14	15	16	18	20	24
Elbow 90° long	9.5	11	12	16	20	24	28	30	32	36	40	48
Elbow 90° short	14	16	18	24	30	36	42	45	48	54	60	72
Elbow 45°	7.2	8.4	9	12	15	18	21	23	24	27	30	36
Elbow 22½°	4.8	5.5	6	8	10	12	14	15	16	18	20	24
Gate valve	9.6	11	12	16	20	24	28	30	32	36	40	48
Flap valve	19	22	24	32	40	48	56	60	64	72	80	96
Pontoon connection	12	14	15	20	25	30	35	38	40	45	50	60

TABLE 72. - Characteristics of dredge pipe

Internal diameter, inches	Standard lap welded pipe: Thickness, inch	Weight per foot, pounds	Bursting strength, lb./sq.in.	Spiral pipe: Thickness, U.S. gage	Weight per foot, pounds	Theoretical safe working head, feet	Bursting strength, lbs./sq.in.
6	0.28	19.0	1/1,000	16	5.3	400	1,250
8	.322	28.5	1/1,000	16	7.1	346	935
10	.366	40.5	1/1,000	16	8.8	276	750
12	.375	49.5	3,150	16	10.6	230	625
15	.375	65.0	2,500	14	17.0	230	625
16	.375	69.4	2,340	14	18.1	216	585
18	.375	77.8	2,082	14	19.9	192	525
20	.5	116.2	2,500	14	22.1	172	470
24	.5	138.0	2,082	12	36.5	201	540
26	.5	150.0	1,922	12	39.5	184	505
30	.5	172.0	1,666	10	56.8	208	560

1/ Routine factory test.

A choice between the use of spiral or standard pipe will depend largely upon the experience of the operator. Standard pipe of equal diameter, as will be noted, is much heavier; it will also withstand much higher heads or pressures, but spiral pipe is strong enough for average dredging requirements.

Standard pipe, on the other hand, will withstand internal abrasion and wear longer than spiral pipe. Where discharge lines are carried over land or can be held in a fixed position it is sometimes possible to revolve them one third or one half and thus prolong their life against abrasion from sand and gravel dragging along the bottom.

Where it is necessary to submerge the pipe line to the river or lake bottom as a protection against swift currents, storms, or surface traffic, the pontoons may be filled with water and sunk. When it is necessary to move the line, the water is blown out with compressed air and the line floated.

Capacity

The capacity of a pump and pipe-line system depends upon the internal diameter of the pipe, the velocity of flow, and the percentage of solids in the mixture.

The diameter of the discharge line is ordinarily fixed by the discharge diameter of the pump.

The velocity of flow in pumping sand and gravel will seldom be less than 10 feet per second or more than 18 feet. The usual velocity averages 12 feet per second.

The amount of solids in the mixture is largely a function of the skill of the operator and the character of the gravel. It ranges from 5 to 20 percent by volume and usually averages 10 to 15 percent.

Table 73 gives the capacity of pump and pipe-line systems for the average range of velocities and percentages of solids. The figures in the table are based on sand and gravel weighing 100 pounds per cubic foot, loose measure. The capacity corresponding to heavier material will vary in direct proportion; hence, for sand and gravel weighing 110 pounds per cubic foot the figures should be increased 10 percent.

TABLE 73. - Capacity of pump and pipe-line systems, tons per hour[1]

Pipe diameter, inches	Solids, percent	Velocity of flow, feet per second: 10	11	12	13	14	15	16	17	18
6	5	17.7	19.5	21.2	23.0	24.7	26.5	28.4	30.0	31.8
	10	35.4	39.0	42.3	46.0	49.5	58.2	56.7	60.0	63.6
	15	53.1	58.5	63.5	69.0	74.2	79.7	85.0	90.0	95.4
	20	70.8	78.0	84.7	92.0	99.0	106	113	120	127
8	5	31.3	34.5	37.7	40.8	43.8	46.9	50.0	53.3	56.4
	10	62.7	69.0	75.4	81.6	87.6	93.9	100	107	113
	15	94.1	104	113	122	131	141	150	160	169
	20	125	138	151	163	175	188	200	213	225
10	5	49.0	54.0	58.8	63.7	68.7	73.6	78.4	83.4	88.3
	10	98.0	108	118	127	137	147	157	167	177
	15	147	162	176	191	206	221	235	250	265
	20	196	216	235	255	275	295	314	333	353
12	5	70.7	77.7	84.8	91.8	98.8	106	113	120	127
	10	141	155	170	184	198	212	226	240	254
	15	212	233	254	276	296	318	339	360	382
	20	283	311	339	367	395	424	452	480	509
14	5	96.3	106	115	125	135	144	154	163	173
	10	193	212	231	250	269	288	308	327	346
	15	289	318	346	375	404	433	462	490	520
	20	385	424	462	500	538	577	616	654	693
16	5	126	138	151	163	176	188	201	214	226
	10	251	277	302	327	352	377	402	427	453
	15	377	415	453	490	528	565	603	641	679
	20	503	553	604	654	704	754	804	855	905
18	5	159	175	191	207	223	239	255	271	286
	10	318	350	382	414	446	478	510	541	573
	15	477	525	573	621	669	716	764	812	860
	20	637	700	764	828	892	956	1,018	1,082	1,145
20	5	197	216	236	256	275	295	315	334	354
	10	394	433	472	512	551	590	630	669	708
	15	590	650	708	768	826	885	945	1,000	1,060
	20	787	866	945	1,023	1,100	1,180	1,260	1,338	1,415
24	5	283	311	339	368	396	424	452	481	509
	10	566	622	679	735	782	848	904	962	1,018
	15	849	933	1,017	1,100	1,187	1,272	1,356	1,442	1,527
	20	1,130	1,244	1,357	1,470	1,583	1,696	1,810	1,923	2,036
30	5	330	363	397	430	463	496	528	562	594
	10	660	727	794	860	925	992	1,057	1,124	1,188
	15	990	1,090	1,190	1,290	1,388	1,487	1,585	1,686	1,783
	20	1,320	1,453	1,587	1,720	1,850	1,983	2,113	2,248	2,380

1/ In this table sand and gravel is assumed to weigh 100 pounds per cubic foot. For material weighing 110 pounds per cubic foot increase figures 10 percent, etc.

To illustrate the method of calculating the equipment necessary for a gravel pumping plant and pipe-line transportation an assumed set of conditions is selected and the equipment calculated in example 14.

Example 14

It is assumed that the gravel deposit contains considerable clay requiring careful washing and that it is mined by hydraulicking and the gravel sluiced to a sump in the pit. The distance from sump to washing plant is 4,000 feet, and the top of the plant is 40 feet above the water level in the sump. The desired rate of production is 3,000 tons per 10-hour day. It is further assumed that the gravel particles are water-worn to rounded edges and that there are no troublesome boulders which cannot be eliminated by the hydraulic mining method. Because of the clay it is decided to pump the gravel from sump to washing plant.

As an initial calculation, as discussed in part 3 of this report under hydraulic dredges, the capacity of a dredge pump in cubic yards per hour when pumping 10 percent solids at a velocity of 12 feet per second is seven eighths the square of the diameter of the pump in inches. The desired capacity in this example is 3,000 tons, or 2,000 cubic yards per 10 hours, or 200 cubic yards per hour; then

$$200 = \frac{7}{8} d^2 ,$$

$$d = 15.1.$$

Therefore a 16-inch pump is indicated.

The total head against which the unit must operate is as follows:

	Feet
Static head	40
Friction head (4.7 to 6.6 feet per 100 feet of 16-inch pipe at 12 feet per second[7/]) at average 5.6 x 4,000	224
Suction head assumed at	15
Total hydraulic head	279

Since 150 feet is the customary limit for a standard pump it is evident that a booster pump is necessary in the line. If the pipe line is constructed so that the grade is uniform from sump to discharge and the booster pump is in the center of the line, the head on each pump will be as follows:

7/ Table 32, part 3 of this series, chapter on "Hydraulic Dredges."

Sump pump:	Feet
Suction head	15
Friction head at 5.6 x 2000 ft.	112
Static head	20
Positive head delivery to booster	12
Total	159

Booster pump:	
Friction head at 5.6 x 2000 ft.	112
Static head	20
	132
Less positive head from sump pump	12
Total head	120

To equalize the duty on both pumps the booster is placed 1,700 feet from the sump and 2,300 feet from the plant. The head on each pump is then as follows

Sump pump:	Feet
Suction head	15
Friction head 5.6 x 17	95.2
Static head	17.0
Pressure to booster	12.0
Total	139.2

Booster pump:	
Friction head (5.6 x 23)	128.8
Static head	23.0
	151.8
Less positive head from sump pump	12.0
Total head	139.8

Assuming that the pumps are identical in design and each has a 48-inch impeller, then each should operate at 414 r.p.m. to develop 140 feet of head (table 38, part 3).

The power required for each pump can be calculated from the formula

$$HP = \frac{QWH}{33000}$$

in which Q = quantity of material, in cubic feet per minute;
W = the weight of the material per cubic foot;
H = the total head, in feet.

From table 70 a cubic yard of sand and gravel with 10 percent solids will require 254 cubic feet of water, making a total of 254+27 = 281 cubic feet of mixture per cubic yard of gravel. Since the pumps must handle 200 cubic yards of gravel per hour the quantity of material handled per minute will be

$$\frac{200 \times 281}{60} = 937 \text{ cubic feet.}$$

From table 34, part 3, the specific gravity of a mixture containing 10 percent solids is 1.12, therefore each cubic foot will weigh

$$1.12 \times 62.5 = 70 \text{ pounds.}$$

Substituting in the formula,

$$HP = \frac{937 \times 70 \times 140}{33,000} = 278.2 \text{ (water horsepower).}$$

The motor rating or brake horsepower required on each pump would then be approximately 600 HP.

It will be noted from table 73 that a 16-inch pipe line handling 10 percent solids at a velocity of 12 feet per second will deliver 302 tons per hour, or that required in the example.

In computing the total head for example 14, no allowance has been made for the additional friction due to valves, bends, and flexible couplings in the pipe line. Accurate calculations would require that they be included in computing total head.

TOWBOATS AND BARGES

The previous discussion of haulage units has been confined to their use in intraplant haulage, that is, transfer of material from excavator to treatment plant. Except for motor trucks the types of haulage so far mentioned are not adapted to delivery beyond the plant limits. While towboats and barges are commonly used for intraplant service, they are more often used to move material in interplant haulage, as between the treatment plant on the dredge and the storage yard or other plant on shore or direct to consumer.

Barges moved by towboats or tugs are the most popular type of conveyance serving sand and gravel dredges, and whether their use is confined to intraplant or extended to interplant haulage will depend on whether the dredge carries partial or complete treatment machinery. Dredges of the clamshell and dipper type do not ordinarily have washing and screening equipment aboard, and the barges serving them are restricted to intraplant haul. As previously noted, however, the hydraulic dredge may provide its own conveying system through pipe lines. The ladder dredge, on the other hand, is almost invariably equipped with washing and screening machinery, and the barges serving it are usually employed for interplant haulage. However, the wide geographic range possible with any type of dredge necessitates equipment capable of covering distances greatly in excess of those ordinarily covered by the previously discussed methods. Hence, most barge designs are such that they may be used for either intraplant or interplant haulage. The exceptions are where a dredge is used in an artificial pond which has been so enlarged that the distance is too great for direct pump and pipe-line delivery.

Barges may be the sole vehicle, they may be supplemented by other types of equipment, or they may themselves be the supplementary equipment. For example, the barge may be loaded directly by the dredge and towed to the treatment plant. If the treatment plant is on the shore line the dock equipment used to unload the barge may deliver directly to the plant. If the plant is some distance away, some supplementary form of conveyance is necessary. This may consist either of cars hauled by locomotive, autotrucks, an engine plane operated by a hoist, or a conveyor-belt system. Barges may also be unloaded by crane or other means to underwater storage, from which a hydraulic dredge, power scraper, or slackline cableway excavator may complete the transfer to the treatment plant.

In other instances a hydraulic dredge may excavate and deliver material directly to a centrally located, underwater storage from which barges may be loaded by various means for further movement. Barges are also used frequently for haulage between shore storage yards, between shore plant and scattered storage yards, or from either plant or storage to consumers delivery point.

As most used, sand and gravel barges have no means of generating power either for self-propulsion or for unloading their cargo. Recent development, however, has produced both self-propelled and self-unloading barges, but these developments are as yet confined to localities with specific need for such equipment.

The self-unloading type may be a simple rectangular barge with a crane wielding a clamshell bucket mounted on one end. The barge is loaded by the dredge, towed to its destination by towboat or tug, and left to unload its cargo while the tug proceeds on other business. Another type of self-unloading barge is equipped with conveyor belts feeding a boom conveyor which can be swung over the side to discharge to shore storage or other point as desired.

Self-propelling barges ordinarily are specially designed equipment for specific use or are converted lake or ocean steamers. They may or may not be capable of unloading their own cargo.

Barges without power are more or less standardized equipment. By this it is meant that manufacturers of this type of equipment have standardized on sizes and details of design to the extent that certain sizes and designs are built and carried in stock for quick delivery. Barges equipped with power are built or designed to meet specific conditions and are not standardized.

Structural Limitations

Sand and gravel barges for river or wet-pit service are usually rectangular in shape with vertical sides, and both ends are raked to provide minimum resistance to towing. Either wood or steel is used in their construction, the present tendency being toward steel, as it provides greater strength, less weight, and greater capacity per unit of weight.

Barges used in lake or ocean service are frequently of the same design but may have round bottoms with blunt-tapered ends.

Sand and gravel barges for pit, river, lake, or ocean service are constructed in three general designs, depending upon the position of the load with respect to the deck line. Hopper-type barges (fig. 24) carry the entire cargo below the deck. This type is used less extensively for gravel transfer. It has the advantage of providing a lower center of gravity when loaded and hence is more seaworthy. It has a disadvantage, however, in that the water entering the hopper with the gravel or from rain must be pumped out. Hopper-type barges are somewhat cheaper but are structurally weaker.

What may be termed a "semi-hopper-type barge" is designed so that the load is carried partly below and partly above the deckline. It is subject to the same objection as the full hopper barge in that water entering the hopper must be pumped out. It is less stable in stormy weather but somewhat stronger structurally.

The flush-deck type (fig. 25) of construction is probably used most for gravel haulage. On it the whole cargo is carried in a cargo box built on the barge deck. In the sides of the cargo box limber holes are provided through which rain water and water accompanying the gravel can drain off automatically, obviating the necessity for pumping. The flush-deck barge carries its entire cargo above the water line and is therefore the least seaworthy of the three types. However, it can be demonstrated that a properly designed barge with a hold free of water cannot be capsized with a reasonable load, although in a severe storm it may lose part of its cargo due to waves. If water is allowed to accumulate in the hold this type of barge is not seaworthy, but the accumulation of water in the hold is a matter of maintenance and should be prevented in any type of barge.

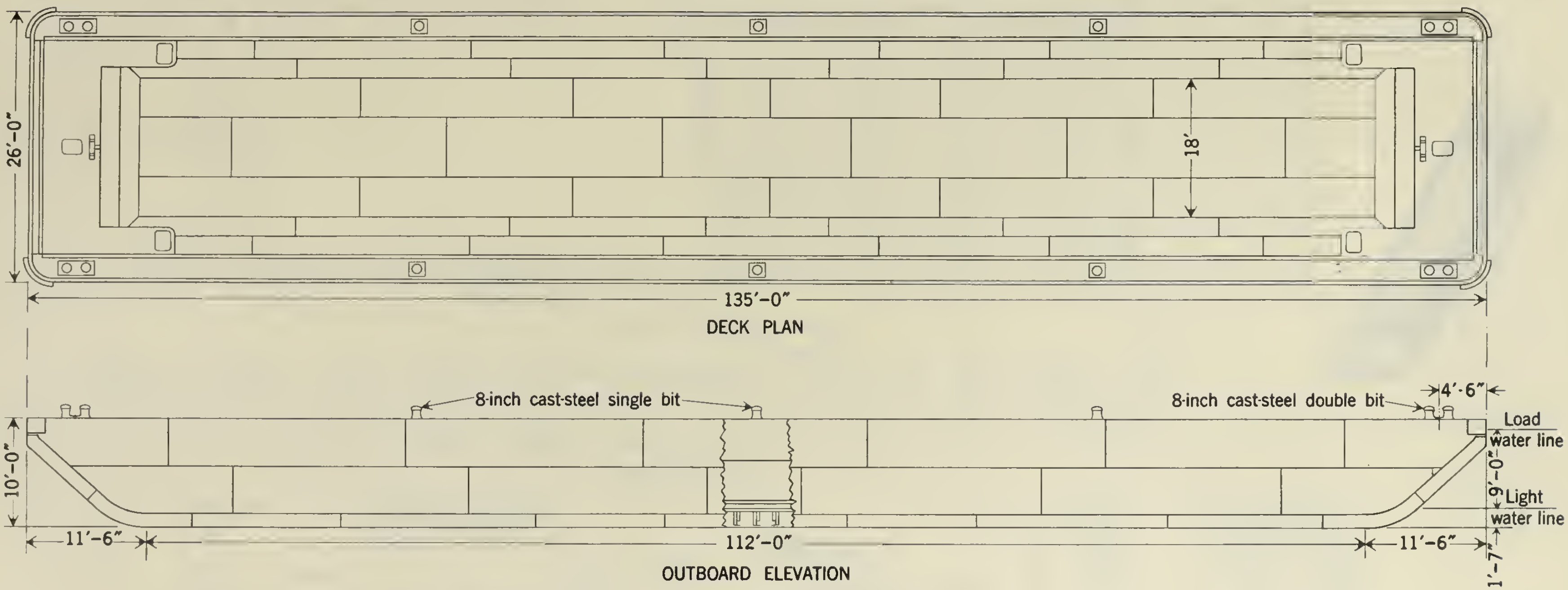
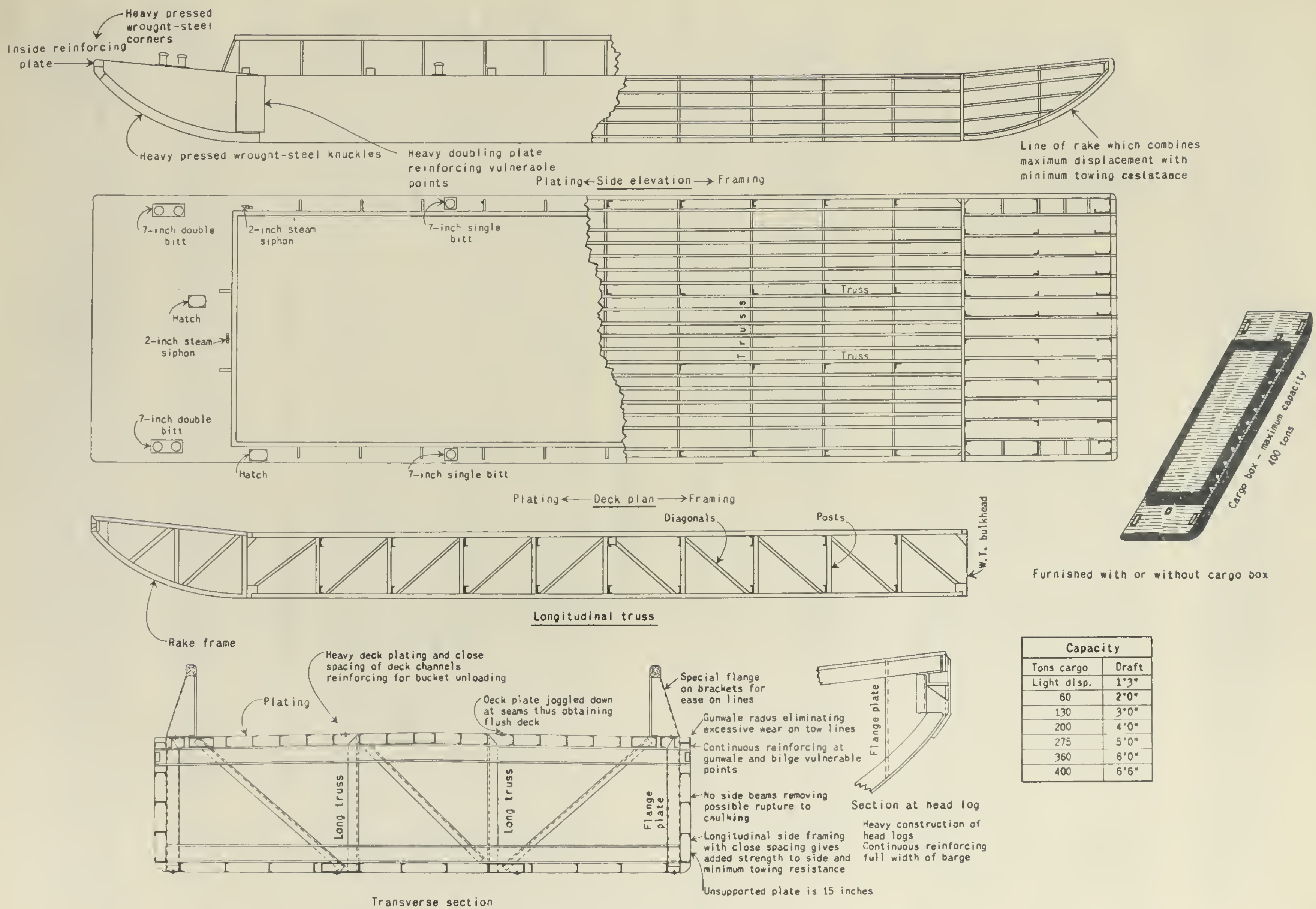


Figure 24.—Plan and elevation of hopper-type barge.



Capacity	
Tons cargo	Draft
Light disp.	1'3"
60	2'0"
130	3'0"
200	4'0"
275	5'0"
360	6'0"
400	6'6"

Figure 25.- Details of flush-deck type of barge.

For river or pit service the flush-deck type of barge is designed for only 6 inches of freeboard when fully loaded, although hopper-type barges require more. For lake or ocean duty more out-of-water height is desirable to withstand waves and storms.

Generally, the size of the barge is increased as the length of the haul increases, because the first cost is less per unit of capacity and large tows are handled easier in large units. Thus, for long hauls handling river freight, barges frequently are 2,000 or 2,500 tons in capacity. For the usual short or gravel-plant haul, however, barges seldom exceed 800 tons in capacity and may be as small as 100 tons. The smaller sizes are usually of wood, but these seldom exceed 100 feet in length or 400 tons in capacity, while many operators find steel the more economical structural material for capacities of 300 tons or more. As an example of the relation of barge size, capacity, and depth of draft, one manufacturer lists the following stock sizes in steel barges.

Length, feet	Beam, feet	Depth, feet	Capacity load, tons	Draft at capacity load, feet
100	26	6.5	365	6.0
130	30	7.5	635	7.0

It is not within the province of this paper to discuss manufacturing details of barge construction.

Auxiliary Equipment

The ordinary nonpower gravel barge requires some means of locomotion, and this is provided by towboats or tugs. Towboats of the side or stern paddle-wheel type (side paddles are now seldom used) are used in river service or wherever shallow drafts are necessary. They have rectangular hulls, raked bow and stern, and towing drafts ranging from 2 to 4 feet. Their towing speed depends upon the size of the tow and whether the loads must be transported against or with the river current. While a towboat may be capable of developing a speed of 12 to 15 miles or more per hour by itself in still water, this may be cut to 2 to 8 miles by the size of the tow or the river current. In fact, a towboat may be able to handle several barges without loads upstream, whereas one loaded barge would tax its power under the same conditions.

Side or stern paddle-wheel towboats are suited only to river or pit haulage.

For lake or ocean service, where shallow draft is unnecessary, tugs with round bottoms and screw propellers are usually employed to provide motive power.

Steam is probably the most popular type of power used, although gasoline-driven internal-combustion engines are common in the smaller sizes of towboats and tugs. Diesel-powered boats and tugs are a more recent development, and their use is increasing as old equipment is replaced.

The design and construction of towboats and tugs is a matter of special marine-engineering study, and this type of construction is not standardized. Usually each unit is built to specification for a particular service, although many boats, built for one service, have been used for others either without change or after remodeling to suit the new conditions. Their design requires such special knowledge that the sand and gravel operator will do best to leave this to reputable builders, contenting himself with stipulating the details of the service for which he desires the unit.

Self-propelled sand and gravel barges are really marine equipment specially designed for the sand and gravel trade, and their design and construction will depend entirely upon the service in which they are to be employed.

Self-unloading barges are equipped with various types of unloading machinery. The simple barge with crane and clamshell bucket mounted on one end has been mentioned. Conveyor belts are commonly used to deliver the cargo to shore or other floating equipment. To adapt the conveyor belt to this use the belt itself is usually mounted on a structural-steel boom capable of radial movement over the side of the barge. The boom belt may be loaded by power scrapers operating in the hold of the barge and delivering to a hopper over the inboard end of the conveyor, or it may be loaded by a belt conveyor taking its load from chutes in the bottom of the cargo hoppers. Other types of self-unloading barges omit the radial-boom conveyor and simply extend the hopper conveyor over the bow of the barge. This type of barge is moored with its bow to the dock, and the conveyor usually delivers to a hopper over another conveyor on the dock. In such barges the flush-deck cargo box is replaced by hoppers supported by steel framework above the deck on which the conveyor operates. The barge may generate its own power to operate the belt or may depend upon power lines on the dock to supply current to the conveyor motors. The Ward Sand & Gravel Co.[8/] for wet-pit operation uses a barge (fig. 26) of this design in which the customary hull is replaced by 8 to 12 cylindrical steel pontoons 10-1/2 feet in diameter by 40 feet long. Such construction is of course adapted only to the quiet water encountered in pit operation.

Capacity

The capacity of a towboat and barge system of haulage depends upon a number of factors so variable that tables are impossible. The factors may be summarized briefly as the length of haul, the loaded and empty speed, the time required to load and unload barges, and the idle time due to waiting for towboats. The length of haul will of course vary with the position of the dredge with respect to the unloading point. For pit operations this will probably remain fairly constant but will increase gradually as the dredge advances away from the plant. For river operations the length of haul will be constant while the dredge operates over one bar. This may require a few days or the whole operating season depending on the size of the deposit. Where several bars containing materials of different physical characteristics

8/ Ward, F. L., Methods and Costs of Mining and preparing Sand and Gravel at the plant of the Ward Sand & Gravel Co., Oxford, Mich.: Inf. Circ. 6580, Bureau of Mines, 1932, 16 pp.

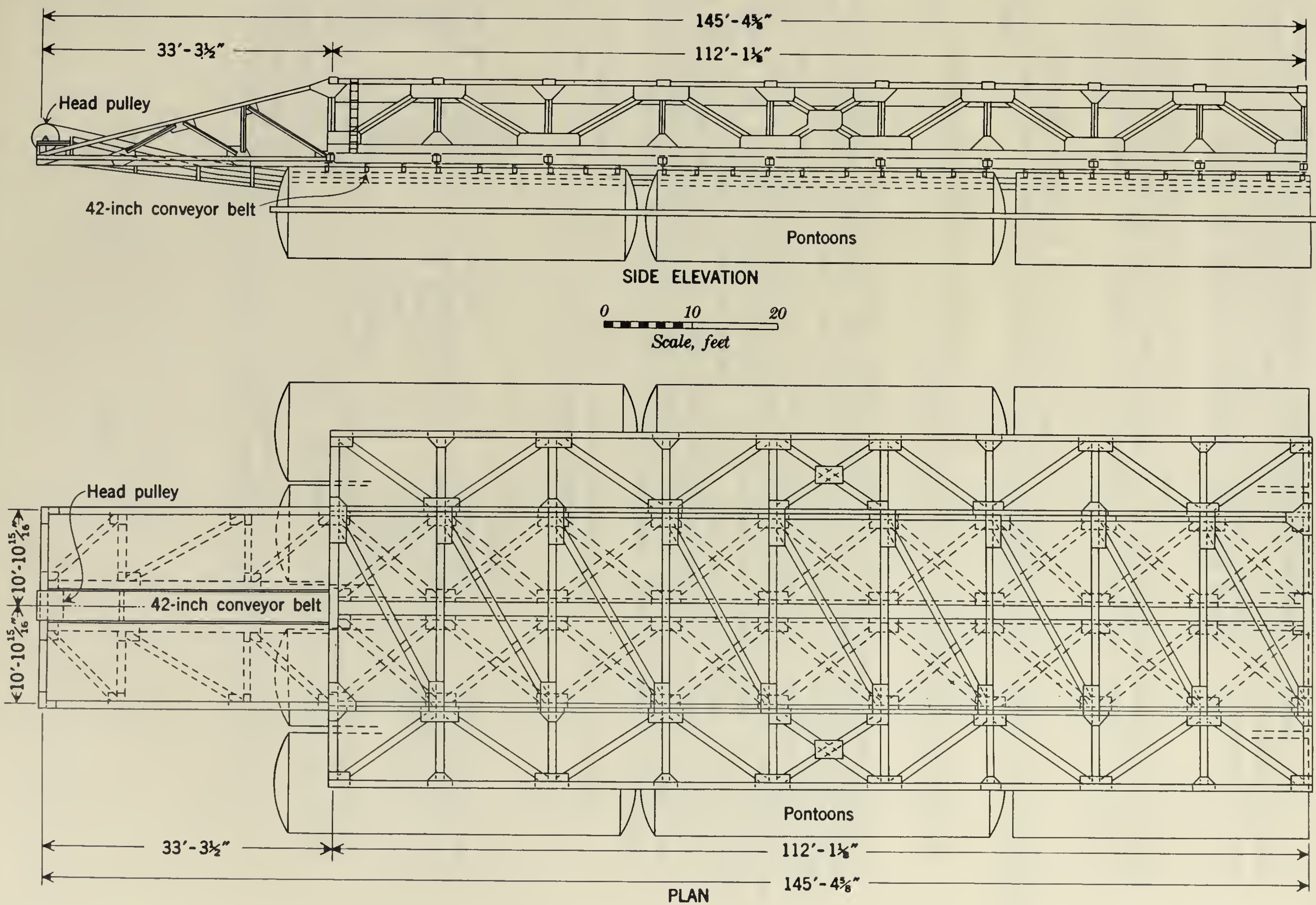


Figure 26.—Self-unloading barge, Ward Sand & Gravel Co.

are to be operated with one dredge it may be necessary to move the dredge frequently and thus vary the haulage distance. Again, the dredge may be stationary, but it may be necessary to deliver material to several storage or delivery points at unequal distances from the dredge.

The loaded and empty speeds will depend upon the power of the towboat and whether the barges are moved in quiet water or with or against a river or tidal current. The speeds will depend also upon the number of barges in the tow.

The time required to load the barge will depend upon the type of deposit being excavated, the type and size of the dredge, and the skill of the dredge runner.

The unloading time will depend upon the type of unloading equipment used.

The idle time will depend largely upon the efficiency of management, provided adequate equipment is available. If the number of barges or towboats is insufficient the time required to deliver a tow and return an empty may be more than that required by the dredge to load the corresponding tonnage. In that case barges must remain idle at the dredge while awaiting the towboat. If the unloading equipment is inadequate the barges will be idle at that end waiting to be unloaded. Since the loading and unloading times are themselves variable and the traveling time is also variable owing to different haulage distances, towboats and barges are probably subject to a greater percentage of unavoidable idle time than other types of haulage.

As examples of the diversity in the requirements found necessary for barge haul, three typical installations follow.

Number of barges	Barge capacity, tons	Dredge no. and type	Dredge capacity, tons per hour	Haulage distance, miles
20 1/	650	2 ladder	900	2 to 30
3 2/	3,000	1 pump	1600	$\frac{1}{2}$
48 3/	600	1 ladder	500	35

1/ Duffy, J. H., Method and Cost of Dredging Sand and Gravel by the Ohio River Sand Co.: Inf. Circ. 6421, Bureau of Mines, 1931, 17 pp.

2/ See footnote 8.

3/ Williamson, G. H., Sand and Gravel Dredging Methods and Costs of J. K. Davison & Bro.: Inf. Circ. 6582, Bureau of Mines, 1932, 10 pp.

AERIAL TRAMWAY

The aerial-tramway system of haulage, also known as a "ropeway", "wire-rope conveyor", or "cableway",[9] is a method of transporting material a few hundred feet or many miles by carrier buckets suspended in the air from moving or standing wire ropes which in turn are supported by wooden or steel towers.

There are several distinct types of aerial trams which have been developed by differences in local haulage conditions. The following are the principal types now in use:

1. Single-rope, fixed-clip tramway.
2. Monocable, or single-rope, saddle-clip tramway.
3. Bicable or double-rope tramway.
4. Jig-back or two-bucket, two-track, reversing tramway.
5. To-and-fro or single-bucket, single-track, reversing tramway.
6. Hoisting and transporting cableway.

In the first two types the buckets are attached to and suspended from endless moving ropes passing over sheaves on the supporting towers.

In the other four types the buckets are suspended from, and move over, standing track cables supported by towers and are attached to a traction rope which provides motion.

The single-rope, fixed-clip tramway consists of an endless moving rope to which the buckets are fixed at regular intervals. It is designed to run at low speeds carrying light loads. As the buckets are always attached to the running rope they must be loaded and unloaded while in motion.

The monocable, or single-rope, saddle-clip tramway consists of an endless moving rope to which the buckets are secured by friction grips built into the design of the saddle clip. It is designed for high speeds (up to 500 feet per minute) and heavy loads (up to 250 tons per hour). The bucket carriage is supplied with sheaves permitting movement over stationary rails and a V-shaped saddle clamp which grips the running cable securely. The buckets are detached automatically for loading and unloading and reattached automatically for despatch.

The bicable or double-rope tramway consists of an endless moving rope to which the buckets are manually or automatically attached and detached by friction clamps of various designs. The buckets are attached to carriages hung from sheaves which travel over standing track cables supported by towers. They are designed for high speeds and large individual bucket capacities. Loaded buckets are attached to the traction rope at specified intervals and are carried over one track cable to their destination, where they are usually detached for dumping and then refastened to the traction rope for return to the loading point over a second parallel track cable. Under favorable conditions it is unnecessary to detach the bucket for dumping. The buckets are attached to the running rope manually or automatically.

9/ A "cableway" strictly speaking, is a special form of tramway and should not be loosely classed as an aerial tram.

Under favorable conditions the buckets may be fixed to the traction rope and remain fixed during both loading and dumping. When so constructed special apparatus must be used to load the bucket while it is in motion. The traveling speed is necessarily slower and seldom exceeds 300 feet per minute.

The jig-back or two-bucket reversing tramway is similar in general operation to the bicable tramway, except that only two buckets are used, one on each track cable, and the running rope instead of traveling continuously in one direction is reversed at each trip. Thus, each bucket travels back and forth on its own track cable. This type is designed for slow or high speeds and light or heavy loads and is sometimes operated without mechanical power similar to an inclined plane - the descending loaded bucket furnishing power to pull back the ascending empty. Each bucket is attached to one of two hoist ropes with a tail rope running from one bucket to the other. The jig back can also be used to haul loads up an incline, but mechanical power must be applied.

The to-and-fro or single-bucket reversing tramway resembles the jig back but has only one bucket running on a single track cable. The direction of travel of the traction rope is reversed each trip. The bucket is attached to one end of a hoist rope and gravity is depended upon for movement in one direction. The system is usually designed for comparatively light loads and moderate speeds but may be built for heavy loads. At times a tail rope is necessary.

The hoisting and transporting cableway is a to-and-fro single-carriage system but is so designed that the carriage may be stopped and held in any position on the track cable and the bucket lowered and hoisted from that point. It is designed for low or high speeds but for heavy single loads (up to 150 tons or more). The aerial tram transports but does not hoist loads, whereas the cableway does both. As mentioned, it is rather a specialized type of aerial conveyance.

The monocable (single-rope, saddle-clip) and the bicable are the two types used most for haulage under conditions required at sand and gravel plants. Either type is essentially an endless-rope haulage system in which the graded roadway, rails, and ties of the surface system are replaced by track cables supported on towers.

The great advantage of the aerial tram over other systems is that it requires no prepared roadway on the ground. This avoids interference with surface operations along the right-of-way and traffic at intersections with highways. This advantage is emphasized in country of rough topography and in the crossing of rivers or other bodies of water. By locating a tower on either or both banks of a river the aerial tram carries its load across on suspended cables, where a surface system would require a bridge. If the stream is navigable the surface haulage problem is further complicated by the necessity of a swing or lift bridge to permit shipping to pass. With the aerial tram haulage is simplified by increasing the height of the towers on the river bank so that the buckets pass high enough to avoid water-borne

traffic. In rough topography the aerial tram proceeds in a direct line between the loading and discharge points and the haulage distance is therefore a minimum. Since the tram can negotiate favorable or adverse grades any difference in elevation between terminals causes no operating difficulty. If the grade is against the loads more power must be expended, of course, but this would be necessary with any other system of haulage between the two points. To keep within permissible haulage grades over rough topography the surface, systems, on the other hand, must either resort to heavy grading in the roadway, requiring deep cuts and high fills, or, when the difference in elevation between terminals is too great for permissible grades on a straight line, to a longer curved roadway or possibly switchbacks, all of which require more or less grading.

Like barge and truck loading systems the aerial tram may be used for either short intraplant service or for long interplant haulage. However, barge and truck systems are more flexible in that they can transfer material from one or more points to one or more other points due to their independence of a single roadway. The aerial tram, on the other hand, is confined to transfer of material from one fixed point to another, although buckets may be loaded or discharged from more than one point along the line.

The aerial-tram system compares favorably with other types of haulage in installation cost over level topography and ordinarily has considerable advantage in this respect in rugged country. Any type of power may be used, including steam or internal-combustion engines, electric motor, or line shaft. Modern installations usually employ the electric motor. A technical discussion of the application and electrical hook-up for different types of motors, as applied to aerial trams, will be found on page 419, volume 133 (1932) of Engineering and Mining Journal.

When loads are transported over level surface or adverse grades by aerial tram, power must be expended as with any other type of haulage. When the material is lowered down steep grades, as often occurs in mountainous country, the tram may be operated as a gravity plane in which the descending loads return the empty buckets without mechanical power except that of the brake drum. With some such installations electric motors are installed for use when no loads are descending. When the tram is in operation the motor acts as a brake and generates electric power for use elsewhere. Sometimes both favorable and adverse grades are necessary. The driving motor may then be used part time to operate the line and part time as a power generator. In long lines it is customary to install a starting motor in addition to the regular operating drive. The duty of the starting motor is to overcome the initial inertia of the buckets and haulage rope and start them moving. When once in motion the starting motor is cut out and the regular drive operates the line.

Structural Limitations

The possibilities and advantages of the aerial tram for haulage in rugged, mountainous, or otherwise inaccessible country have long been recognized by metal miners, but its successful competition with other systems in

the simpler forms of plant haulage has progressed slowly. So far there have been few applications to sand and gravel exploitation, although recent improvements in design and construction have solved many earlier operating problems.

As previously stated the aerial tram is essentially an endless-rope haulage system in which the carrying vehicles are transported through the air instead of on the ground. To accomplish this the rails and ties of the surface system are replaced by track cables and supporting towers.

In the bicable type there are two track cables, one of larger diameter carrying the loaded buckets in one direction and the other of smaller section carrying the returning empty buckets. The track cables are anchored securely at one end and stretched over metal saddles on the towers to a tension station at the other end. In the tension tower the cables are attached to heavy weights which provide the power tension to enable the cable to support a load of similar weight suspended between towers and the weight of the traveling buckets. Tension towers may be placed at either the loading or discharge terminal on short hauls or in the middle or at other convenient intermediate points for long hauls. When the tension tower is placed in the line the buckets are transferred from one cable to the next over steel rails and rail points, thus permitting continuous operation without stopping. Track cables are made in various designs. In some the core is a round strand of wire surrounded by interlocking wires, some or all of which are specially shaped for two reasons: (1) To present a smooth surface, free from impact, for the passage of the carrier wheels, and (2) to prevent exterior broken wires from protruding or unwinding and thus avoid entanglement or derailment of the carrier. In some installations a lang-lay haulage rope is used for track cable. The locked-coil cable is more expensive but presents less friction to the wheels of the bucket carriage and usually has a longer life. Haulage rope is often used for short hauls or for temporary construction.

Track cables pass over the towers and are supported in cast-iron or steel saddles which are grooved to fit the underside of the cable but offer no obstruction to the passage of bucket wheels. The cable rests in the saddle groove and is free to move longitudinally in either direction. This is necessary, as constant shifting of the load due to the traveling buckets causes constant change in the arc of sag between towers and hence continual movement of the track cable in the saddle grooves.

A third, or haulage, rope, usually of lang-lay construction, transmits motion to the buckets. This rope is ordinarily driven by a bullwheel operating on a vertical shaft and equipped with multiple finger grips or leather lining to provide driving friction. The haulage rope may be below or above the track cables, depending upon the design of the tram. The buckets are attached to it by friction grips. These grips may be operated manually by an attendant at the loading station, or they may be so constructed as to clamp automatically to the haulage rope as the bucket is pushed onto the line. In either case the bucket is detached automatically at the discharge terminal. The bucket, in passing from one cable to another at an intermediate tension station, remains fastened to the haulage rope as it passes over the rail connection between track cables.

In the monocable type the traction or haulage rope of the bicable system is eliminated, and the heavy track cables themselves provide motion. They pass over a series of sheaves at each tower arranged so as to minimize the arc of sag at that point. No intermediate tension stations are required, but the proper tension is applied to the cables at one or both of the terminal stations. The tower sheaves are so designed that the bucket carriage passes over them without releasing the friction grip in the saddle clips. At the dumping terminal it is unnecessary to detach the bucket as must be done in a bicable system. The incoming cable is led over a sheave, at the terminal and directly alongside is placed the graded end of a steel rail. As the bucket approaches the terminal the carriage sheaves ride the rail and lift the saddle clips clear of the cable, thus detaching the bucket and transferring it to the rail where it is moved manually to the dumping position.

Aerial-tram towers may be either of wood or steel, depending upon local economic factors. They are short or long depending upon the contour of the surface, which also determines whether their spacing along the line shall be at equal distances or irregular intervals. If spaced at short distances less tension is needed on the track cables, but the cost for tower construction is greater. Close spacing also causes less deflection in the track cables as the loaded buckets approach and leave a tower saddle. This results in less wear on the cable and hence longer life. Tower spacing must be designed carefully to provide an equable balance between initial installation cost and maintenance expense. Where the tram passes over highways, rivers, or other bodies of water the towers on either side are usually higher than at other points to provide clearance for highway or water-borne traffic. Where the tram spans rivers or valleys the distance from bank to bank or hill to hill often far exceeds the customary tower spacing. Long spans require greater tension on the track cables and necessitate greater sag in the suspended cables. This increased sag may require increased tower height, even though there is no traffic interference in the span. Single spans of this nature are frequently necessary in mountainous country and have been operated successfully over distances approaching a mile in length. In crossing highways it is customary to suspend a steel-wire network below the tram buckets as a safety precaution against dropping of material on the highway.

Although longitudinal movement of the track cables is provided for in the saddle grooves no provision is made for movement of the cables at right angles to the direction of bucket travel. The weight of the cable and buckets normally prevents the cable from moving out of the groove. Under abnormal conditions, however, such as excessive or unequal loading, the track cable may be lifted vertically out of the saddle. In that case, if the towers are not alined carefully, the cable when it drops back will settle either on the tower cap inside the saddle or outside the end of the cap. In either case the next bucket passing is apt to catch at the tower or drag on the ground and cause trouble. For this reason aerial trams are built without horizontal curves in most installations, and great care is necessary to assure that the tower saddles are in exact alinement. Modern design, however, has devised mechanical appliances which permit considerable curvature in the line, but a special tower must be erected at the point of the curve to accomplish this.

The usual method is to pass the track cables around a deflecting sheave of large diameter and bypass the buckets around the curve from one tangent to the other by means of steel rails on the curve tower. This requires an attendant at the station to reclamp the buckets to the haulage rope after the latter has passed its deflecting sheave, as it is difficult to devise a clamp that will retain its grip while passing around such a sheave.

The aerial tram can be constructed to transport material over any sort of surface and for any reasonable distance. It may be utilized to transfer material from one building to another a few hundred feet apart, each building functioning as a terminal with an unsupported span between buildings, or it may function as a haulage unit many miles long. For example, the Peru Mining & Smelting Co. employs an aerial tram to haul ore from its mines to the seaboard more than 30 miles away. However, these long trams are not built in one self-contained operating unit. They are usually divided into independent units 1 to 6 or 7 miles long, each unit delivering to the next one either through a transfer tower containing storage bins or by passing the buckets from one section to the next. Each section has its individual power drive. This division into sections also permits an angular change of direction with minimum difficulty.

The average sand and gravel operator, however, would not be as interested in a long haul as in a single self-contained unit handling his material from pit to plant or from plant to transportation terminal.

Aerial-tram buckets vary in design. Some are carried on axles similar to surface cars and travel above the track cables. With such buckets two track cables carry the loads and two return the empties. The wheels on the ends of the axles are grooved to fit the cables, and the haulage rope is above the track cables. In the more commonly used design a bucket is hung in a structural-steel carriage or frame supported from a pin in a 2- or 4-wheel trolley running on a single track cable. The bucket hangs below the track cables, and tha haulage rope is attached to a clamp on the supporting frame. Each tower also carries idler rolls which carry the sag of the haulage rope between buckets. Both designs have their advocates and critics, but the latter seems to be more popular, judging from the number of installations.

Aerial trams are not suited to handling material direct from excavators. Being tied to a fixed loading point the tram cannot follow the excavator. Some other form of haulage which is flexible must intervene between excavator and tram.

Aerial-tram buckets are comparatively small units and consequently are not suitable for handling large lumps or boulders. The efficiency of a tram depends largely on its ability to keep a large number of small-unit carriers in constant motion. This necessitates quick loading and dumping. Material containing a large percentage of boulders presents obvious difficulties to quick loading, hence the tram operates more efficiently with crushed or small material that can be loaded quickly and smoothly through chutes from bins.

Bucket sizes vary with the individual installation, depending upon the capacity desired, the physical character of the material handled, the rope speed of the tramway, and the spacing of the buckets on the line. Trams have been operated successfully with buckets having capacities ranging from 500 pounds to 4 tons each. A mine in Germany using autotrucks to take material from the excavator has designed the truck body so that it can be picked up by a sling on the aerial tram and used as a tram bucket. This arrangement avoids the necessity for crushing machinery and loading bins between the excavator and the tram.

Table 74 has been compiled from articles in the technical press to show the characteristics and range of aerial tramways.

As stated previously, aerial-tram buckets are usually loaded from bins through chutes. Loading bins serving conveyor belts cannot be made portable owing to mechanical difficulties involved in varying the length of the haul. Loading stations can be inserted, however, at various points along the tram, hence loading is not confined to one point.

Tram buckets can be dumped by hand or by automatic devices at any point along the line. Usually buckets are dumped into receiving bins, but sometimes the tram is used to deliver to ground storage or waste dumps. The dumping point may then be at a tower or between two towers as desired. In the latter case automatic dumping devices are fastened to the track cables at the desired point.

Auxiliary Machinery

If the aerial tram is considered as a system consisting of towers, track cables, haulage rope, traveling buckets, and power supply, the only auxiliary machinery required is that for loading and dumping the buckets. This usually involves bins at both loading and discharge terminals. Loading bins are filled by some other haulage unit, such as a conveyor belt, power scraper, or cableway excavator. If the excavated material contains large pieces or boulders it is customary to install a crusher over the loading bin or at some point in the flow of material preceding the bin. In installations such as that mentioned in Germany loading bins and crushers are eliminated, but there are few examples of this type of installation.

Since the tram is tied to a stationary loading point material must be brought to it by other haulage systems; thus they may be considered as service equipment.

At the discharge terminal a bin is commonly provided over which the buckets may be operated by hand or by the haulage rope and dumped at any point desired either manually or automatically. Usually buckets must be detached from the haulage rope at this point as at curve stations. Discharge bins may serve as temporary storage to feed the washing or treatment plant or may be merely intermediate points in the haul supplied by one section of the aerial tram and loading another section or some other type of haulage such as railway cars or autotrucks.

TABLE 74. - Aerial-tram installations

Tram no.	Haulage distance, feet	Bucket capacity, pounds	Number of buckets	Bucket spacing Ft.	Bucket spacing Sec.	Haulage speed, ft. per min.	Track cable diameter, inches load	Track cable diameter, inches empty	Haulage rope diameter inches
1 1/	31,050	896	---	200	29	420	1	1	---
2 2/	1,200	1,000	5	---	--	---	-	-	---
3 3/	26,400	1,200	100	---	--	---	1 3/8	1	5/8
4a 4/	10,162	2,000	---	200	--	500	1 1/2	7/8	5/8
4b	5,150	3,800	---	228	27.4	500	1 3/4	1	3/4
5 5/	2,500	2,000	16	320	48	400	1 1/2	7/8	5/8
6 6/	1,655	2,400	21	324	--	350	1 3/4	1 1/4	3/4
7 7/	6,671	3,000	88	157	--	450	1 5/8	1	1
8 8/	3,000	---	---	---	--	---	-	-	---
9 9/	6,600	4,000	80	---	--	---	-	-	3/4
10a 10/	740	8,000	2	---	--	---	2 1/4	2 1/4	7/8 & 3/4
10b	1,060	8,000	2	---	--	---	2 1/4	2 1/4	7/8 & 3/4
11a 11/	21,921	---	---	---	--	550	1 3/8	1 1/8	3/4
11b	9,000	---	---	---	--	500	1 1/8	1	3/4
11c	35,265	---	---	---	--	550	1 3/8	1 1/8	3/4
11d	30,560	---	---	---	--	550	1 3/8	1 1/8	3/4
11e	31,413	---	---	---	--	550	1 3/8	1 1/8	3/4
11f	16,008	---	---	---	--	550	1 3/8	1 1/8	3/4
11g	17,778	---	---	---	--	550	1 3/8	1 1/8	3/4

Tram no.	Longest clear span, feet	Angles No.	Angles Degrees	Difference in elevation of terminal, feet	Capacity, tons per hour	Power, HP Start	Power, HP Run
1 1/	2,925	1	40	0	60		70
2 2/	---	0		-220	15		5
3 3/	---	-	---	---	25		50
4a 4/	---	2	(144 (170	+33	150		50
4b	---	0		+8	250		30
5 5/	---	0		0	75		--
6 6/	450	0		+110	135		30
7 7/		0		(+420 (-215	260	30	10
8 8/	---	0		-200	180		25
9 9/	750	0		0	---		60
10a 10/	740	0		-357	150		0
10a	1,200	0		-642	150		0
11a 11/	---			+8	12.5		40
11b	---			-559	50		30
11c	---			-1,202	15	40	75
11d	---			-2,063	15	40	75
11e	---			+780	15	20	40
11f	---			(-5,321	15	75	
11g	---				15	75	

1/Indian Copper Corporation, Min. Mag., vol. 50, no. 6, June 1934, p. 340. 2/Clearwater Lime Products Co., Orofino, Ida., Pit & Quarry, vol. 19, no. 12, Mar. 12, 1930, p. 67. 3/Northwest Magnesite Co., Pit & Quarry, vol. 19, no. 12, Mar. 12, 1930, p. 51. 4/Penn Dixie Cement Corporation, Pit & Quarry, vol. 27, no. 4, October 1934, p. 35. 5/Rock Products, vol. 36, no. 8, August 1933, p. 27. 6/Ross Island Sand & Gravel Co., Rock Products, vol. 33, no. 2, Jan. 18, 1930, p. 25. 7/Superior Portland Cement Corporation, Pit & Quarry, vol. 20, no. 2, Apr. 23, 1930, p. 41. 8/Basalt Rock Co., Napa, Calif., Pit & Quarry, vol. 18, no. 9, July 31, 1929, p. 35. 9/U.S. Gypsum Co., Alabaster, Mich., Rock Products, vol. 33, no. 19, Sept. 13, 1930, p. 105. 10/Olympic Portland Cement Co., Pit & Quarry, vol. 19, no. 12, Mar. 12, 1930, p. 41. 11/Peru Mining & Smelting Co., Eng. & Min. Jour., vol. 134, 1933, p. 287.

Capacity

The capacity of an aerial tram depends upon the size of the individual buckets, their spacing on the haulage rope, their speed of travel, and the method of loading them.

For the same haulage distance the capacity can be increased as follows: (1) By substituting an equal number of larger buckets, (2) by using more buckets of the same size but decreasing the distance between them, or (3) by increasing the traveling speed.

For an increased haulage distance the same capacity may be maintained by simply increasing the number of buckets on the line, the spacing and speed remaining constant. The length of haul therefore affects capacity only if a limited number of buckets is available.

Within certain mechanical limits any or all three of these methods can be used to increase capacity.

Often the size of the buckets can be increased without changing the size of the track cables, but before this is attempted care must be exercised to see that the increased concentration of load between any two towers does not cause the track cable to be lifted clear of the saddle on the next tower. This is apt to occur when the tram extends over a long, level or fairly uniform ascending or descending route or where loading and discharge terminals are at higher elevations than intervening towers.

The same care must be exercised in increasing the number of buckets by closer spacing, even though the bucket size is not increased. Decreasing the spacing may cause two loads between towers where formerly only one was necessary. This doubles the traveling load on that particular span, increases the sag in the track cables, and tends to lift the latter from the saddles in succeeding towers.

In either instance a reduction in the tension on the track cables may keep them in their saddles, if sufficient clearance above ground is available, but in so doing the sag is increased between all towers, and the deflection at each tower is increased as the buckets pass over it, thus increasing cable wear due to movement in the saddle and to flexure. Usually this evil is corrected by increasing the height of the tower from which the track cable is being lifted.

Increasing capacity either by larger buckets or more buckets of the same size may require greater tension on the track cables, and this may cause them to lift off the saddles on low towers. Either method may also necessitate replacing the track cables with others of larger diameter, which in turn may require a redesign of the trolley wheels on the bucket carriage.

Capacity can often be increased by increasing haulage speed, but this entails careful study of the driving mechanism, particularly with respect to friction on the drive wheel and the setting of the bucket clamps. Increased speed requires greater driving tension and tighter bucket clamps.

The capacity of an aerial tram is directly affected by the efficiency of the loading device. When it is loaded from bins the chute and gate must be quick-acting, and the material must flow easily and smoothly through the chute to give greatest efficiency. If the material contains boulders or clay rapid loading is difficult owing to blocking in the chutes. When material of this type is hauled by aerial tram it is customary to enlarge the loading station so that surplus empty buckets are always available for loading from more than one chute. A timing device is also helpful so that the operator may send the loaded buckets away at regular intervals. Timing devices may be run by gears from the main shaft and ring a gong at the proper interval, or markers may be placed on the haulage rope at the required distance. The former device is more satisfactory and more generally used.

Owing to the variation in length, traveling speed, and other local conditions it is impracticable to tabulate aerial-tram capacities based on either bucket size or speed. Capacity must be computed for each installation according to local conditions. When all factors are known capacity can be computed accurately if no correction is necessary to account for partly-filled buckets. Since tram buckets usually are not loaded automatically but by manpower, calculation of capacity must take into account the personal equation, as with other types of equipment, hence working capacity is always less than theoretical capacity. The difference will depend upon the efficiency of the loading device and the loader.

The theoretical capacity of an aerial tram may be computed in numerous ways; the following is probably as simple as any.

Let

C = theoretical capacity of the tram, in tons per hour;
W = rated load capacity of each bucket, in pounds;
N = number of buckets delivered per hour

Then

$$C = \frac{WN}{2,000}$$

Let

v = rated volume of each bucket, in cubic feet, and
m = average weight of a cubic foot of the material carried.

Then

$$W = vm$$

Let

t = time interval between buckets, in seconds;
d = distance interval between buckets, in feet;
s = speed of the haulage rope, in feet per minute.

Then

$$N = \frac{3,600}{t} = \frac{60s}{d}$$

and by combination in the original formulas

$$C = \frac{WN}{2,000} = \frac{vmN}{2,000} = \frac{9vm}{5t} = \frac{3vms}{100d}$$

The formula used will depend upon the form in which the known data are presented.

Example 14

An aerial tram is assumed to have been installed with the following operating conditions, and it is desired to know its capacity when handling sand and gravel.

The speed of the haulage rope (s) is 450 feet per minute.

Each bucket has a capacity (v) of 9 cubic feet.

The buckets are spaced (d) on the line at 180 feet.

Sand and gravel ranges in weight from 2,900 to 3,240 pounds per cubic yard, or 107.4 to 120 pounds per cubic foot.

Therefore m will range from 107.4 to 120 pounds.

Then in the formula,

$$C = \frac{3vms}{100d},$$

C will range from

$$\frac{3 \times 9 \times 107.4 \times 450}{100 \times 180}$$

or 72.5 tons per hour at the minimum value of m to

$$\frac{3 \times 9 \times 120 \times 450}{100 \times 180}$$

or 81.0 tons per hour at the maximum value of m.

In example 14 it was assumed that a tram was already set up, and the calculation was made to determine merely its capacity when handling sand and gravel. Field calculations usually involve a different approach to capacity problems. The operator wants to know the details of a tram that will deliver a certain quantity of material from one point to another in a given time. Example 15 illustrates this approach.

Example 15

An operator wishes to erect a tram that will deliver his material, which averages 110 pounds per cubic foot (m), from pit to plant at the rate of 120 tons per hour. Then

$$120 \text{ tons per hour} = \frac{120 \text{x} 2{,}000}{60} = 4{,}000 \text{ pounds per minute.}$$

With a rope speed of 450 feet per minute (s) and buckets spaced to arrive at 30-second intervals (t), the distance spacing, (d), is

$$\frac{450}{2} = 225 \text{ feet.}$$

Individual bucket loads are

$$\frac{4000}{2} = 2{,}000 \text{ pounds.}$$

This size of the buckets (v) is

$$\frac{2{,}000}{110} = 18.2 \text{ cubic feet,}$$

but to prevent spillage, 20-cubic foot buckets are selected.

Then by formula

$$C = \frac{3vms}{100d} = \frac{3\text{x}18.2\text{x}110\text{x}450}{100\text{x}225} = 120 \text{ tons per hour.}$$

The theoretical capacity is

$$\frac{3\text{x}20\text{x}110\text{x}450}{100\text{x}225} = 132 \text{ tons per hour.}$$

With the same rope speed but a 20-second delivery interval the distance spacing would be

$$\frac{450}{3} = 150 \text{ feet.}$$

Individual bucketloads would be

$$\frac{4,000}{3} = 1,333 \text{ pounds,}$$

and each bucket would require

$$\frac{1,333}{110} = 12.1 \text{ cubic feet}$$

with, say, 15-cubic foot buckets installed.

The working capacity would then be

$$C = \frac{3x12.1x110x450}{100x150} = 120 \text{ tons per hour,}$$

and theoretical capacity would be

$$C = \frac{3x15x110x450}{100x150} = 148 \text{ tons per hour.}$$

Similarly, various other bucket sizes and line spacings could be selected.

Power Calculations

The power required to operate an aerial tram may have as its source the force of gravity, a mechanical motor, or both.

Power is consumed in (1) moving the buckets and their loads horizontally, (2) elevating buckets and loads vertically, (3) overcoming resistance offered by line friction, (4) overcoming resistance offered by friction in terminal machinery, and (5) overcoming inertia in starting and accelerating the buckets to line speed.

If the design of the tramway is such that the material transported must also be elevated, then the installed motor must have enough power to do all these things.

If the tramway is designed to lower material, then the force of gravity may be sufficient to overcome all power-consuming factors and may even develop a surplus for use elsewhere, but with low grades gravity is seldom enough, and additional power is necessary even though the loads move downhill.

Various authorities have compiled formulas for computing the power required to operate an aerial tram or the power developed by it[10]. The object

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10/ Peele, Robt., Mining Engineer's Handbook: John Wiley & Sons, 2d ed., New York, 1927, pp. 1745 - 1788.

being to calculate the size of the power unit necessary for economical operation. Formulas differ owing to different methods of calculation or different mechanical data. Consequently, the power requirement found by using one formula may differ from that found by another, even though the basic data are the same. This difference may or may not be serious, but usually analysis will show that the variation is due to the use of different coefficients of frictional resistance.

Frictional resistance is encountered in aerial trams in various forms. In a bicable system the bucket-carrier wheels are subjected to rolling resistance or friction as they travel over the track cables. This will vary with the type of construction used in the track cables and the tension to which they are subjected. Locked steel track cables with smooth surfaces present less rolling friction than haulage or hoist ropes used as track cables. Taut cables offer less friction than slack cables. The bucket-carrier wheels also offer journal friction in their axles, and the traction rope encounters frictional resistance in being dragged over supporting idler sheaves.

Monocable tramways must overcome rolling friction as the rope passes over the tower sheaves, which also offer journal friction.

Both types of trams must overcome frictional resistance in moving the terminal machinery. Terminal resistance or friction will vary with the tension applied to the traction rope. Wire ropes are designed to operate at what is called "working tension", which is 16 to 20 percent of their breaking strength, depending on whether a factor of safety of 6 or 5 is used. Terminal tension can be computed on the basis of the breaking strength of the traction rope. In the bicable system the traction rope performs one function only - bucket movement - hence the tension need be no more than that required to move the loaded buckets and provide sufficient friction at the drive wheel. In the monocable system the running rope performs two functions; it moves the buckets and supports them between towers. It must therefore be under greater tension and sometimes produces greater terminal friction than is required by the traction rope of a bicable system. (However, the increased power cost due to higher terminal friction may be offset by the lower maintenance cost of the simpler monocable system.)

Since plain, roller, or ball bearings may be used in both carriage and tower sheaves and in some terminal machinery, obviously the coefficient of friction will vary considerably according to the construction details.

It is customary to combine the effect of rolling and journal friction in carriage wheels and tower sheaves and represent both by one coefficient designated as "line friction." The coefficient for line friction as given by various authorities ranges from 0.0067 $\left(\frac{1}{150}\right)$ 11/ to 0.03 $\left(\frac{3}{100}\right)$ 12/. The lower figure represents the best modern equipment, including roller bearings.

11/ Information given the author by manufacturers.
12/ Lay, Douglas, Design of Aerial Tramways: Eng. and Min. Jour., vol. 109, June 19, 1920, p. 1359.

The higher figure probably represents plain bearings and may even include a surplus to cover terminal friction.

The coefficient for terminal friction as given by various authorities ranges from 0.00556 $(\frac{1}{180})$[13] to 0.015 $(\frac{1}{67})$[14].

For preliminary estimates the coefficient of line friction may be taken as ranging from 0.007 for antifriction bearings to 0.02 for plain bearings and from 0.005 to 0.015 for terminal bearings.

In using any coefficient, however, allowance must always be made for inefficient lubrication, which may totally destroy the value of any coefficient and the accuracy of computations.

In view of the preceding discussion the horsepower required to operate an aerial tram hauling loads up-grade or the power developed when the loads move down-grade may be computed from the following formulas.

$$HP = \frac{PV}{33,000}$$

in which

HP = horsepower, applied or produced;
P = net pull on the traction rope, pounds;
V = speed of the traction rope, feet per minute.

$$P = T - S + p$$

in which

T = pull on the taut side, in pounds;
S = pull on the slack side, in pounds;
p = pull due to terminal friction.

$$T = WH \pm FWL$$ [15]

$$S = wH \pm FwL$$ [16]

in which

W = weight per foot on the loaded side, in pounds;
w = weight per foot on the empty side, in pounds;
H = difference in elevation between terminals, in feet;
F = coefficient of line friction;
L = horizontal length of the tram, in feet.

$$W = \frac{g+e}{d} + r$$

$$w = \frac{e}{d} + r$$

13/ Information given the author by manufacturers.
14/ Peele, Robt., Mining Engineer's Handbook: John Wiley & Sons, 2d ed. New York, 1927, p. 1773. Marks, Lionel S., Mechanical Engineer's Handbook: McGraw-Hill Book Co., 3d ed. New York, 1930, p. 1419.
15/ Sign of the last term is "plus" when the rope is pulled up-grade and "minus" when it is let down.
16/ See footnote 15.

in which

g[17] = weight of the load in one bucket, in pounds;
e = weight of one empty bucket and carriage, in pounds;
d = distance between buckets, in feet;
r = weight of 1 foot of traction rope, in pounds.
$L = \sqrt{D^2 - H^2}$,

in which D = the inclined length of the tram, in feet

$$p = 0.8\ Bf,$$

in which

B = breaking strength of the traction rope, in pounds, and
f = coefficient of terminal friction.

Combining and substituting the above in one formula:

For loads up-grade,

$$HP = \frac{\left[(WH + FWL) - (wH - FwL) + 0.8\ Bf \quad V\right]}{33{,}000}$$

For loads down-grade,

$$HP = \frac{\left[(WH - FWL) - (wH + FwL) - 0.8\ Bf \quad V\right]}{33{,}000}$$

The preceding formulas assume that the size and weight of the traction rope are known, but none attempts to compute what that size and weight should be.

The size of the traction rope depends upon the total tension to which it will be subjected. This total tension must not be confused with rope pull, although it is based on it. Traction ropes are subject to the same laws as any band drive and may therefore be calculated from the power formulas for band drives.

The pull transmitted to the line by the traction rope depends upon the friction between the rope and the surface of the bull wheel or driving drum. The driving wheel or drum may be a steel sheave with unlined rope groove; the groove may be lined with wood, rubber, or leather; or the sheave may be fitted with self-gripping fingers. In any case, the pull it can exert depends on friction, which, in turn, depends on the tension applied to the slack side of the sheave. By analytical mechanics the relation between the tension on taut and slack sides to prevent slipping is expressed in the following formula:

17/ In calculating the starting pull g is usually doubled.

$$T - S_t e^{f\pi n}$$

in which

T = taut-side tension, in pounds;
S_t = slack-side tension, in pounds;
e = base of Napierian logarithms = 2.71828;
π = 3.1416,
f = coefficient of friction between rope and sheave;
n = number of half laps (180°) on the sheave.

$$T-S = S_t(e^{f\pi m}-1)$$

from which the slack-side tension necessary to prevent slipping may be computed and the total rope tension and size of traction rope.

For convenience the values of f and $e^{f\pi n}$ for the most common conditions are given in table 75.

TABLE 75. - Values of f and $e^{f\pi n}$

Number of half laps (n)	Condition of surfaces	Type of sheave					
		Plain, unlined		Wood-lined		Leather-or rubber-lined	
		f	$e^{f\pi n}$	f	$e^{f\pi n}$	f	$e^{f\pi n}$
1	Wet	0.08	1.3	0.17	1.7	0.40	3.5
1	Dry1/	.17	1.7	.23	2.1	.49	4.7
3	Wet	.08	2.2	.17	4.9	.40	43.4
3	Dry1/	.17	4.9	.23	9.2	.49	106.2

1/ Figures for wet surfaces are customarily used.

Bucket and carriage weights are not standardized but vary with the design of different manufacturers. There is no definite relationship between weight and size. Table 76 shows the approximate range in size and weight.

TABLE 76. - Approximate weight of aerial-tram bucket and carriage

Size, cubic feet		Weight, pounds	
From	To	From	To
4	6	300	375
7	15	400	800
16	25	550	1,000

Example 16

The conditions are assumed to be the same as those used in example 15. The average weight of the sand and gravel was 110 pounds per cubic foot, the size of the buckets was 20 cubic feet, the distance between buckets was 225 feet, the speed of the traction rope was 450 feet per minute, and the load in one bucket was 2,000 pounds.

Additional assumptions are as follows: The inclined distance between terminals is 3,000 feet, the discharge terminal is 100 feet above the loading terminal, plain bearings are used with a line-friction coefficient of 0.02, plain bearings are used with a terminal-friction coefficient of 0.015, and bucket and carriage are assumed to weigh 800 pounds.

The size of the traction rope depends upon the pull required, the calculation of which involves the weight and therefore the size of the traction rope. Calculation must then be a matter of "cut and try."

With a bicable system and the loads pulled up-grade the weight per foot, if the rope weight is neglected, becomes:

$$W = \frac{g+e}{d} = \frac{2,000 + 800}{225} = 12.4 \text{ pounds}$$

$$w = \frac{e}{d} = \frac{800}{225} = 3.55 \text{ pounds.}$$

The inclined distance between terminals is given as 3,000 feet.

$$L = \sqrt{D^2 - H^2} = \sqrt{3,000^2 - 100^2} = 2,998.33 \text{ feet.}$$

The ratio of difference in elevation or lift to length is so small in this instance that it is disregarded, and 3,000 feet is taken as the horizontal length of the tram. With a steep incline, however, this ratio will be appreciable and must be accounted for in the calculation.

Substitution of the values of W and L in the equations for the pull on the taut and slack sides gives

	Pounds
$T = WH + FWL = (12.4 \times 100) + (0.02 \times 12.4 \times 3,000) =$	1,984
$S = wH - FwL = (3.55 \times 100) - (0.02 \times 3.55 \times 3,000) =$	142
$T - S =$	1,842

If a plain, unlined sheave is used to drive with three half laps then the slack-side tension is

$$T\text{-}S = S_t(e^{f\pi n} - 1)$$
$$1,842 = S_t(2.2-1)$$
$$S_t = \frac{1,842}{1.2} = 1,535 \text{ pounds.}$$

Since the taut-side pull was 1,984 pounds and the tension required on the slack side is 1,535 pounds, then by addition the total tension in the rope is 3,519 pounds; thus

$$3,519 \times 5 = 17,595 \text{ pounds} = 8.8 \text{ tons breaking stress.}$$

From manufacturers' catalogs a 6 by 7 cast-steel rope 9/16 inch in diameter has a breaking stress of 9.4 tons and weighs 0.48 pound per foot, whereas the same type of rope $\frac{1}{2}$ inch in diameter has a breaking stress of 7.5 tons. Since the breaking stress of the $\frac{1}{2}$-inch rope is less than that required, the 9/16-inch size is chosen. Recalculating,

$$W = \frac{2,000 + 800}{225} + 0.48 = 12.88 \text{ pounds}$$

$$w = \frac{800}{225} + 0.48 = 4.03 \text{ pounds.}$$

		Pounds
T =	(12.88 x 100) + (0.02 x 12.88 x 3,000) =	2,061
S =	(4.03 x 100) - (0.02 x 4.03 x 3,000) =	161
T-S =		1,900

$$1,900 = S_t(2.2-1),\ S_t = \frac{1,900}{1.2} = 1,583$$
2,061 + 1,583 3,644 pounds total tension.
3,644 x 5 = 18,220 pounds = 9.1 tons breaking stress.

This is below the tabulated strength of the 9/16-inch rope selected.

$$p = 0.8\ Bf = 0.8 \times 18,800 \times 0.015 = 226 \text{ pounds}$$
$$P = T-S + p = 2,061 - 161 + 219 \doteq 2,126 \text{ pounds.}$$

$$HP = \frac{PV}{33,000} = \frac{2,126 \times 450}{33,000} = 29$$

The motor rating would be 25 percent higher, making the required motor 36.25 HP, the nearest commercial size to which would be 40 HP.

Incidentally, since S or the line pull on the slack side is 161 pounds and the slack-side tension necessary is 1,583 pounds, the tension weight necessary to provide this tension is

$$2\ (1,583 - 161) = 2,844 \text{ pounds.}$$

Assuming the same conditions except that the best antifriction bearings on both line and terminal are used, the trial calculation for rope size would be

		Pounds
T =	(12.4 x 100) + (0.007 x 12.4 x 3,000) =	1500
S =	(3.55 x 100) - (0.007 x 3.55 x 3,000) =	280
T - S =		1220

If a leather-lined sheave wheel is used with one half lap, then

$$1,220 = S_t(3.5 - 1)$$
$$S_t = \frac{1,220}{2.5} = 488$$

1,500 + 488 = 1,988 pounds total rope tension
1,988 x 5 = 9,940 pounds = 5 tons breaking stress.

This requires a 7/16-inch, 6 by 7, cast-steel rope weighing 0.29 pound per foot and having a breaking stress of 5.8 tons.

Recalculating,

$$W = \frac{2,000 + 800}{225} + 0.29 = 12.69 \text{ pounds.}$$

$$w = \frac{800}{225} + 0.29 = 3.84 \text{ pounds.}$$

$$\begin{array}{lr} & \underline{\text{Pounds}} \\ T = (12.7 \times 100) + (0.007 \times 12.7 \times 3,000) = & 1,537 \\ S = (3.8 \times 100) - (0.007 \times 3.8 \times 3,000) = & \underline{300} \\ T-S = & 1,237 \end{array}$$

$$1,237 = S_t(3.5 - 1)$$

$$S_t = \frac{1,237}{2.5} = 495 \text{ pounds.}$$

1,537 + 495 = 2,032 pounds total rope tension.

2,032 x 5 = 10,160 pounds = 5.08 tons, which is below the tabulated strength of the 7/16-inch rope selected.

$$p = 0.8\ Bf = 0.8 \times 11,600 \times 0.005 = 46.4 \text{ pounds.}$$

$$P = 1,537 - 300 + 46.4 = 1,283 \text{ pounds}$$

$$HP = \frac{1,283 \times 450}{33,000} = 17.5$$

If 25 percent is added the motor rating is 22 HP. The tension weight in this case would be 2(495 - 300) = 390 pounds.

If a monocable system is used instead of the bicable the traction rope must have strength enough to support the loads over the longest span without excessive sag or deflection in the line. The permissible sag may be limited by local surface profile or economical tower height. Line sag or deflection depends upon the tension applied to the traction rope, the length and inclination of the span, the weight of the rope, the weight of the buckets and their loads, and the distance between buckets. Tramway design permits a maximum sag of 2½ to 10 percent of the span, but it usually does not exceed 5 percent. Sag or deflection must be calculated for each position of the bucket or buckets on each span and involves formulas for catenary or parabolic curves and higher mathematics.18/ For final tram design the deflection should be accurately calculated from these formulas by an engineer familiar with this class of construction.

18/ Carstarphen, F. C., A Simple Method of Computing Deflections of a Cable Span Carrying Multiple Loads Evenly Spaced: Trans. Am. Soc. Civ. Eng., Paper 1,454, vol. 83, 1919-20, pp. 1383-1408.

Calculation of the power required for a monocable system involves the tension required in the traction rope rather than the sag. However, tension depends upon rope weight, span length, and weight of the loads carried, all of which are involved in computing sag. Therefore, if the maximum sag is assumed the formula for sag may be used to determine the tension required and hence the size of rope necessary.

The tension in an unloaded rope may be found from the following formula:

$$Te = \frac{W s^2}{8 d},$$

and the tension due to a single load at the center of a span, the ends of which are at the same elevation, by

$$Tg = \frac{P s}{4 d}.$$

The total tension on the rope is then Te + Tg. This will be modified when the ends of the span are at unequal elevations. In these formulas,

Te = horizontal tension due to empty rope, in pounds;
Tg = horizontal tension due to a single load, in pounds;
W = weight of the rope per foot, in pounds;
s = length of the span, in feet;
P = gross weight of one bucket and load, in pounds;
d = sag or deflection at the center of the span, in feet.

The greater deflection will be caused by the single load in the center of the span. If a ratio of sag to span of 5 percent and a maximum span of 200 feet are assumed, the tension due to load is

$$Tg = \frac{P s}{4 d} = \frac{2,800 \times 200}{4 \times 10} = 14,000 \text{ pounds.}$$

This may for the moment be assumed as the working tension of the traction rope. Its breaking strength will then be

$$14,000 \times 5 = 70,000 \text{ pounds,}$$

or 35 tons. Reference to manufacturers' catalogs shows that a 6 by 19 special plow-steel rope of 1-inch diameter has a breaking strength of 42 tons and weighs 1.6 pounds per foot.

Recalculation gives

$$Te = \frac{w\ s^2}{8\ d} = \frac{1.6x(200)^2}{8x10} = \underline{\ \ 800\ \ }\ \text{Pounds}$$

$$Tg = \frac{P\ s}{4\ d} = \frac{2,800x200}{4x10} = \underline{14,000}$$

$$Te + Tg = \qquad 14,800$$

$$14,800 \times 5 = 74,000 \text{ pounds}$$

or 37 tons, as the breaking strength necessary, indicating a 1-inch rope.

Substitution in the power formulas gives

$$W = \frac{g+e}{d} + r = \frac{2,000 + 800}{225} + 1.6 = 14.0 \text{ pounds,}$$

$$w = \frac{e}{d} + r = \frac{800}{225} + 1.6 = 5.1 \text{ pounds}$$

$$T = (14.0 \times 100) + (0.02 \times 14.0 \times 3,000) = 2,240 \text{ pounds,}$$
$$S = (5.1 \times 100) - (0.02 \times 5.1 \times 3,000) = 204 \text{ pounds,}$$
$$p \doteq 0.8\ Bf = 0.8 \times 84,000 \times 0.015 = 1,008 \text{ pounds,}$$
$$P = T - S + p = 2,240 - 204 + 1,008 = 3,044 \text{ pounds,}$$

$$HP = \frac{PV}{33,000} = \frac{3,044 \times 450}{33,000} = 41.5.$$

The motor rating would be 25 percent higher, making the required motor 50 HP.

With antifriction bearings the theoretical HP required would be 22.2 and the motor rating, 28 HP.

Then

$$T - S = 2,240 - 204 = 2,036$$

With a driving sheave equipped with steel jaws and having a leverage ratio of 1 to 3 the value of $e^{f\pi n}$ will be 2.2, or the same as a plain sheave with 3 half laps. Then

$$2,036 = S_t(2.2-1),$$
$$S_t = 1,700 \text{ pounds.}$$

The tension necessary on the slack side to prevent slipping is then 1,700 pounds. However, the tension necessary to provide the designed line sag was 14,800 pounds, hence slack-side tension for driving may be disregarded.

The tension weight required will be twice the necessary tension, or 14,800 x 2 = 29,600 pounds.

Aerial-Tram Design

This paper might be greatly extended by discussing aerial-tram design, including the proper heights of towers to provide clearance for loaded buckets, the tension to be applied to obtain the required deflection and driving friction, and other technical problems involved. Since, however, the solution of a single problem for a specific installation could not be applied to another because of local differences and because such a solution involves higher mathematics, the author feels that such discussion is beyond the scope of this circular. For such information the reader is referred to the references cited, to technical handbooks, and to the engineering staffs of manufacturers of aerial trams.

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February 1936

Department of the Interior

UNITED STATES BUREAU OF MINES
John W. Finch, Director

INFORMATION CIRCULAR

MINERAL INDUSTRIES SURVEY OF THE UNITED STATES

IDAHO
SHOSHONE COUNTY
COEUR D'ALENE DISTRICT

THE SILVER BELT AND THE SUNSHINE MINE OF THE COEUR D'ALENE DISTRICT

BY

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IDAHO
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THE SILVER BELT AND THE SUNSHINE MINE OF THE COEUR D' ALENE DISTRICT[1]

By C. E. Julihn[2] and F. W. Horton[3]

CONTENTS

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6876."

2 Principal mining engineer, U.S. Bureau of Mines.

3 Mining engineer, U.S. Bureau of Mines.

FOREWARD

The Mineral Industries Survey of the United States was recently initiated by the United States Bureau of Mines. Its personnel consists of field engineers who visit various localities to study and report upon the current state of their mineral industries. Ultimately the reports will constitute a comprehensive mineral survey of the entire country. As they will show the setting and background of mineral enterprises, they may be used to preface or supplement more detailed reports of mining engineers on individual operations. They should likewise prove useful in other ways to operators and investors interested in the development and maintenance of domestic mining, milling, and smelting operations.

This first paper of the series projected by the Mineral Industries Survey is based upon a reconnaissance of prospects, mines, and plants of mineral industries in Shoshone and adjacent counties of Idaho from August to November 1935. It is an advance section of a report that will cover a more extensive area, of which the silver belt of the Coeur d'Alenes is only a part.

John W. Finch

DIRECTOR, U.S. BUREAU OF MINES.

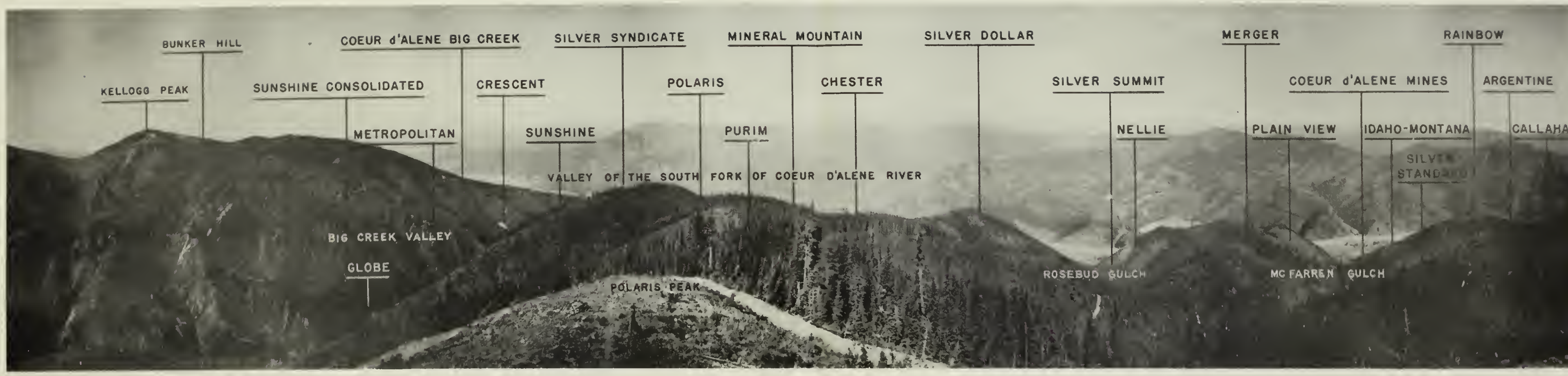

Figure 1.—Silver belt of the Coeur d'Alene district from Polaris Peak.

Figure 2.—Sunshine mine.

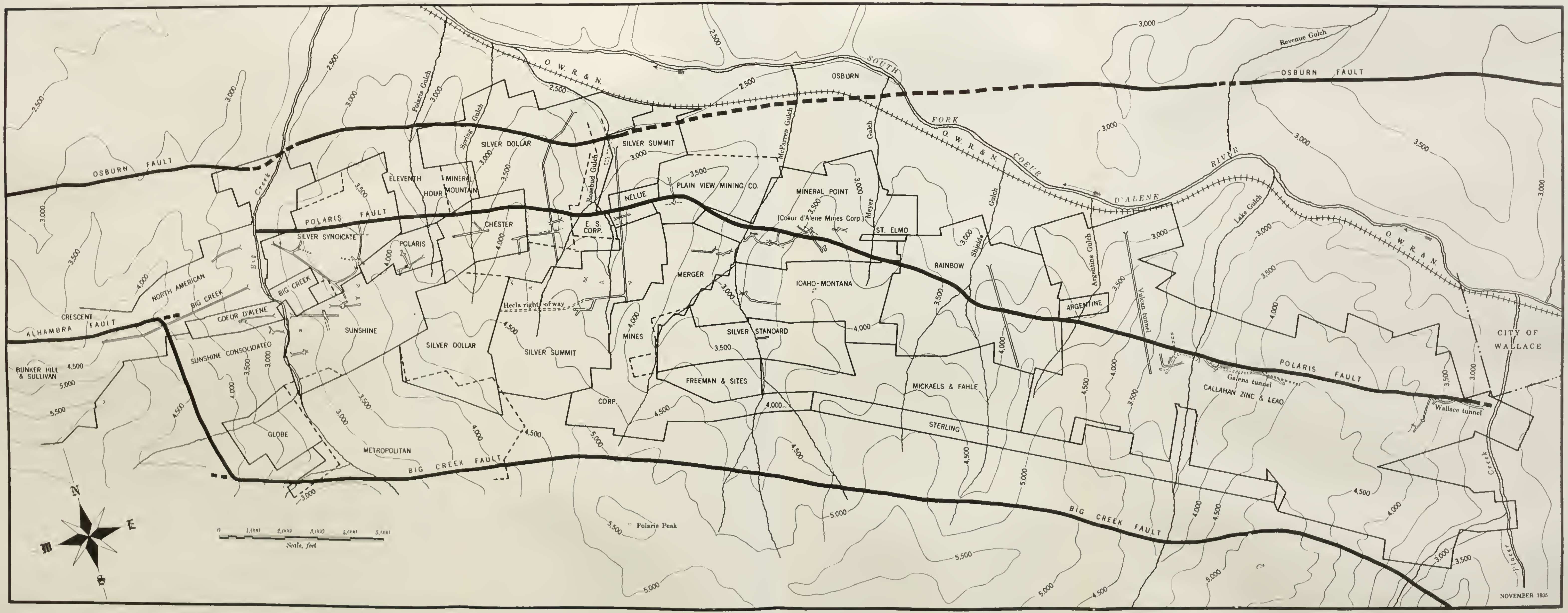

Figure 3.—Map of silver belt of the Coeur d'Alene district.

INTRODUCTION

In the past few years the advance of the Sunshine mine to national prominence as the chief silver-producing mine of the United States has revived interest in the area in which it is situated. Although this area lies between the great lead-zinc-silver mines at Kellogg and those in the vicinity of Burke and Mullan it was generally presumed to be almost barren, as little ore had been found there until substantial ore bodies were developed on deep levels of the Sunshine mine. In early days it was called the "dry belt" because its ores were siliceous and contained little lead; but the classification of ores in smelting as wet or dry, although once customary, has long been obsolete. The area will therefore be referred to in this paper as the silver belt of the Coeur d'Alene district because the value of its ore is due chiefly to silver, the current production of which is large, providing over 60 percent of the silver output of Idaho.

Silver Belt - Its Limits and Character

With a daily production of half a ton of silver or more from the Sunshine mine, the silver belt has become the scene of extensive exploration at many properties, and the outlook for its future seems promising enough to warrant the lively interest now being displayed in the area by the public.

The silver belt includes about 20 square miles of rugged, mountainous country bounded on the morth by the South Fork of the Coeur d'Alene River, the valley of which lies about 3,000 feet below the crest of a steep, wooded ridge about a mile above sea level; the ridge parallels the valley about 2 miles south of it.

The generally recognized western limit of the area is the valley of Big Creek, a stream that flows into the South Fork between Wallace and Kellogg at a point about 3 miles east of the latter. From Big Creek the belt extends east 6 or 7 miles nearly to Wallace between Lake and Placer Creeks. Its width does not exceed 3 or 4 miles at most; some fix the southern limit as the crest of the ridge, although others consider that the belt extends farther south to the crest of the St. Joe Mountains.

The appearance of the area is shown in figure 1, a panorama taken from Polaris Peak. At its left is the valley of Big Creek, where a dump of the Crescent mine is visible. On the east side of the valley, nearly opposite the Crescent, is the Sunshine mine. Its position beyond the mountain in the foreground is indicated, although it cannot actually be seen. It is shown in another panorama with Big Creek, the portal of the adit tunnel, the mill, and other buildings in the foreground and above them the dumps of five earlier tunnels.

Still farther east over a ridge is the Polaris mine, likewise hidden from view, to which the Silver Summit tunnel 2 miles long is being driven from the mouth of Rosebud Gulch.

Through Rosebud and McFarren Gulches may be glimpsed the valley of the South Fork and beyond it a part of the Coeur d'Alene Mountains extending north in the direction of Murray.

The areal relations of the various properties of the silver belt are suggested by figure 3. This map does not attempt to show individual mining claims but only the approximate areas in which they are grouped, subject, of course, to some adjustments of property titles.

ACKNOWLEDGMENTS

The authors were courteously received and generously assisted by all operators of the silver belt. They are especially indebted to Stanley Easton, president and general manager of the Bunker Hill & Sullivan Mining & Concentrating Co.; Roy Hooper, superintendent of the Crescent mine of that company; Frank Eichelberger, vice president and manager of the

Sunshine Mining Co.; J. T. Hall and R. F. Mahoney, of the Sunshine mine staff; J. F. McCarty, president and general manager of tho Hecla Mining Co., which controls the Polaris mine; Harry Pearson, manager of the Silver Summit mine; S. H. Richardson, manager of the Mineral Point mine of the Coeur d'Alene Mines Corporation; and Julius P. Hall, mining engineer and United States deputy mineral surveyor, of Wallace, who granted permission for reproduction of his map of the silver belt.

GEOLOGICAL BACKGROUND

Formations.- The geology of the entire Coeur d'Alene district, including the silver belt, was studied by the United States Geological Survey and the results of the study published in 1908 as Professional Paper 62, "The Geology and Ore Deposits of the Coeur d'Alene District, Idaho" by F. L. Ransome and F. C. Calkins[4]. It shows only four rock formations in the area of the silver belt—the Revett, St. Regis, Wallace, and Striped Peak, all of which are sedimentaries of the pre-Cambrian Belt series. The lowest of these is the Revett, "mainly white, thick-bedded quartzite, in part sericitic." Above it is the St. Regis, "purple and green indurated shales and quartzitic sandstones"; next higher is the Wallace, thin-bedded calcareous shales, with siliceous magnesian limestone and calcareous quartzite in their middle part. The highest of the four is the Striped Peak, "indurated shales, sandstones, and shaly quartzite."

The Wallace occurs over the larger part of the silver belt, the St. Regis being next in extent. The Revett appears only in three small areas, two on the east and another, somewhat larger, at the west of the belt. The Striped Peak tops some parts of the large southerly area of Wallace rocks between the main ridge and the St. Joe Mountains.

Folding, Faulting, and Shearing.- Only the Wallace, the St. Regis, and the Revett occur in the part of the silver belt being most actively explored. This lies between 2 great faults 1 to 1½ miles apart that traverse the belt from west to east. The northern fault is the Polaris, a little less than a mile south of the valley of the South Fork, and the southern is the Big Creek fault; both dip steeply to the south.

As the downthrow side of Polaris fault is to the south, while that of the Big Creek fault is to its north, the block of Wallace and St. Regis rocks between the two has subsided with respect to the rocks flanking it.

This block has been subjected to compressive stresses—evidenced by folding—which probably are also responsible for a zone of shearing roughly parallel both in strike and dip to the two faults described. At the western end of the belt near Big Creek the shear zone is near the middle of the block, about half a mile south of Polaris fault. It strikes a little south of east but farther east gradually bends more to the south until its course is about S. 60° E. It dips steeply to the south and ranges in width from 150 to 600 feet. The ore bodies of the Sunshine mine occur in this zone which might conveniently be termed the "Sunshine shear zone."

On the west side of Big Creek in line with the strike of the shear zone the Alhambra fault is encountered. Some relation between the two may therefore be reasonably inferred. The shear zone may have resulted from an extension of stresses in line with the Alhambra fault diffused through a wider zone; or it may actually be a continuation of the Alhambra fault, rendered obscure by the relatively soft Wallace rocks on both sides of it. The latter seems more probable, as similar shearing, though not so wide, appears to occur to the north of it in the near vicinity of the Polaris fault. For convenience, this zone might be referred to as the Polaris shearing.

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4 See also Ransome, F. L., Ore Deposits of the Coeur d'Alene District, Idaho: U. S. Geol. Survey Bull. 260, 1905, pp. 274-303; Umpleby, J. B., and Jones, E. L. Jr., Geology and Ore Deposits of Shoshone County, Idaho: U. S. Geol. Survey Bull. 732, 1923, 156 pp.

It would be interesting to know whether other shearing of this type is associated with the Big Creek fault, upon which little exploration has been done.

A fourth prominent fault, the Placer Creek, roughly parallels the Big Creek fault half a mile to a mile south of it.

There are also several minor faultings within the silver belt; and the great Osburn fault nearly coincides with its northern boundary, as that fault is followed by the valley of the South Fork.

Vein Minerals.- In the silver belt the predominant vein minerals are siderite and quartz associated with argentiferous tetrahedrite and with galena, pyrite, and occasional chalcopyrite and proustite. The ore of the belt does not differ greatly from some of the silver-lead ores found elsewhere in the Coeur d'Alene district, although it usually contains much less galena and is characterized by the presence of argentiferous tetrahedrite. It may reasonably be supposed to represent merely a phase of the widespread mineralization of the district, determined by such factors as temperature, pressure, and distance from the magmatic source of mineral-bearing solutions.

MINING, EXPLORATION, AND DEVELOPMENT

Only two mines are actually producing silver from the silver belt-the Crescent, belonging to the Bunker Hill & Sullivan Mining & Concentrating Co., and the Sunshine. Both are at the western end of the belt, the Crescent on the west side of Big Creek and the Sunshine on the east side. The former has followed the Alhambra fault for 2 miles from Elk Creek into the valley of Big Creek. Several bodies of silver ore were encountered along the hanging wall of the fault. The ore is concentrated at a mill on Big Creek a short distance below that of the Sunshine mine.

The Sunshine has been mined in some fashion almost continuously since 1884, as its small and uncertain stringers and bunches of ore near the surface proved sufficiently rich to maintain operations. Its large, rich ore bodies were encountered only at considerable depth and so far have tended to improve with depth; the revival of interest in the silver belt as a whole is due largely to this. It is now generally believed that the silver belt is not as barren as was once supposed but that its ores lie in a fairly deep horizon.

East of the Sunshine mine, beyond the crest of a ridge, are the upper tunnels of the Polaris mine which, like the Sunshine, was an early producer of silver from small ore bodies of a shear zone. It is being explored vigorously in two tunnels. One, starting from Polaris Gulch, was driven west along the Polaris vein which lies a little south of Polaris fault. Some characteristic silver-belt ore was found in driving the tunnel and likewise in crosscutting from a winze nearly a thousand feet below the tunnel. The other, the Silver Summit tunnel, is about 1,000 feet lower; it is being driven more than 2 miles from the valley of the South Fork at the mouth of Rosebud Gulch. Its elevation corresponds approximately with that of the main-tunnel level of the Sunshine mine. These extensive operations are sponsored by the Hecla Mining Co., which controls the Polaris mine.

The Silver Dollar Mining Co., formerly the Lincoln, is driving another long tunnel from the valley of the South Fork west of Rosebud Gulch to explore ground east of the areas controlled by the Sunshine and Polaris mines.

The Nellie mine on the east side of Rosebud Gulch shipped silver ore several years ago from workings in the vicinity of the Polaris fault. Its reopening by connection with the Silver Summit tunnel is said to have been proposed. Just east of it is the Plain View, also on the Polaris fault.

In McFarren Gulch, still farther east, are 4 active operations, one, on the old Mineral Point mine, which formerly shipped silver ore mined in the vicinity of the Polaris fault.

Farther east, in Argentine Gulch, is the old Argentine mine, another former producer of silver ore. Beyond it a body of silver ore was also mined in ground of the Callahan group in Lake Gulch.

More detailed accounts of these and other operations of the silver belt follow.

Elk Creek

Alhambra Fault.- Although Elk Creek is west of Big Creek, the generally recognized western limit of the silver belt, attention is called to the fact (see Crescent Mine) that small amounts of silver ore with tetrahedrite were found along the Alhambra fault in the Alhambra tunnel on the Elk Creek side of the Big Creek divide.

Big Creek

Big Creek is by far the largest in volume, length, and area drained of all the numerous creeks of the silver belt flowing into the South Fork of the Coeur d'Alene River. On it are three of the most important properties of the silver belt--the Sunshine, Crescent, and Sunshine Consolidated--and several other properties of considerable interest. It joins the South Fork about 3 miles east of Kellogg.

Sunshine Mine

The Sunshine mine, which has been the largest silver producer in the United States since 1932, is on the east side of Big Creek about 2 miles south of its confluence with the South Fork of the Coeur d'Alene River. It is reached by a good road branching south from the main highway, U. S. No. 10, about 3 miles east of Kellogg. The property centers about the Yankee Boy claim on which silver ore was discovered in 1885 by Dennis and True Blake.

History.- Up to 1899 the Yankee Boy is credited with a production of about $30,000 of shipping ore said to carry 150 to 160 ounces of silver per ton. From 1899 to 1903, inclusive, the property was idle, but during 1904 and 1905 the Blake brothers reported outputs of 7,312 and 5,894 ounces of silver, respectively. During the next 15 years the mine was operated by lessees. No records of its production during the first 6 years of this period are available, but it is known that considerable ore was mined from workings above the No. 4 tunnel. It seems likely that the production before 1912 was about 150,000 ounces of silver.

Production.- In 1921 the Sunshine Mining Co. acquired the property and has steadily continued its development to date, bringing its recorded output to more than 24,000,000 ounces of silver since 1911, as shown by the following table of production.

Production of the Sunshine mine from 1912 to November 1935

Date	Gold ounces	Silver ounces	Copper pounds	Lead pounds
1912	1.11	14,904	2,685	13,896
1913	7.00	67,459	11,032	116,000
1914	9.24	69,039	10,398	90,788
1915	--	30,266	--	77,504
1916	8.06	33,520	8,160	44,742
1917	24.68	126,449	4,928	590,362
1918	6.71	46,136	1,342	178,574
1919	.70	24,145	4,499	55,770
1920	1.24	23,420	3,074	144,197
1921	2.69	53,811	8,486	82,528
1922	11.35	121,572	16,109	139,078
1923	12.06	109,154	14,234	244,089
1924	8.49	186,181	21,888	473,958
1925	1.33	271,200	45,777	703,079
1926	--	406,104	95,918	528,813
1927	--	1,050,507	255,682	1,248,430
1928	--	1,125,280	169,135	1,306,988
1929	--	1,676,412	266,102	1,765,010
1930	--	2,310,845	396,058	1,624,822
1931	2.12	2,409,124	367,815	1,240,866
1932	--	3,015,539	561,159	424,000
1933	50.02	3,127,780	793,984	252,567
1934	108.04	3,456,568	700,000	258,000
1935 (first 10 mos.)		4,660,000	1,067,000	255,000
Total		24,415,415	4,822,355	11,759,062

Vein.- The Sunshine vein strikes approximately east and west and dips 65 to 70° south. The foot wall is distinct, but the hanging wall fades into a sheared zone 400 to 600 feet wide, which roughly parallels the Polaris fault. The country rock comprises shales and quartzites of the Wallace and St. Regis formations, with the lower workings in the latter. The principal vein minerals are siderite and quartz associated with argentiferous tetrahedrite, galena, and pyrite. Some proustite was found in the upper workings. Average ore contains about 45 ounces of silver per ton, although ore running 100 ounces or better is common, and occasionally lenses of almost pure tetrahedrite assaying 1,000 ounces per ton or more are found.

Mine.- The mine is developed by a main adit 1,450 feet long in or adjacent to the vein. The portal of this adit is at an elevation of 2,655 feet. An inclined 2-compartment shaft extends from it to the 1,900-foot level, where a vertical shaft offset from the incline continues to the bottom at the 2,300-foot level. A 4-compartment shaft is being sunk from the surface to connect with a crosscut being driven 450 feet northwest from the vein on the 1,700-foot level. This new shaft is being equipped with a hoist capable of raising 2,000 tons a day from a depth of 4,000 feet. It not only will provide access to the deep levels of the Sunshine but may also serve for the exploration of adjacent properties at depth. Above the main or Sunshine adit are 4 tunnels, the upper 3 of which have been abandoned. Lessees are working ground immediately above the No. 4 tunnel.

Figures 4 and 5 are maps showing the plan and vertical longitudinal section, respectively, of the mine workings.

In the upper tunnels and in several levels below the main-adit tunnel the vein is very irregular in width, both along its strike and dip, ranging from a hardly perceptible seam to as much as 2 or 3 feet. On the 500-foot level below the adit the width increases, and on the 900-foot level it reaches a maximum of 9 feet.

Below the 1,300-foot level the mineralization has much greater regularity. In the lower levels the vein maintains an average width of about 4 feet, ranging usually from 3 to 7 feet. The ore shoot on the 1,900-foot level is about 1,400 feet long.

In November 1935 most of the mine's production came from the 1,500-, 1,700-, 1,900-, and 2,100-foot levels, although some ore was being mined from all levels. Exploration work was being conducted on the 1,900-, 2,100-, and 2,300-foot levels, and the 1,700-foot level was being extended west to explore adjacent ground on the west side of Big Creek. In the lowest level there were excellent showings of ore which appeared to maintain the width, continuity, and values seen in the level above.

Diamond drilling is reported to have proved that the vein continues to a depth of at least 3,300 feet below the main-tunnel level.

A discovery, which may be of great importance, was made in October 1935 in running the crosscut from the 1,700-foot level to the line of the new shaft. A new ore body 5 feet wide, said to average about 30 ounces of silver per ton and 30 percent lead, was encountered. This ore body possibly indicates mineralization at depth of a separate and distinct part of the shear zone.

<u>Mining and Milling Methods</u>.- A flat-back, square-set system of mining is used, and rock fill is obtained from development work. Rill stoping was formerly employed, but at present there are no rill stopes left in the mine.

The mine makes about 40 gallons of water per minute. The hoisting capacity is 500 tons per day. The ore is trammed on the main levels by storage-battery locomotives. Power for both mine and mill is purchased from the Washington Water Power Co.

The ore is broken underground to pass 8-inch grizzlies above the ore pockets. At the surface it is dumped from the mine cars into a coarse-ore bin holding 150 tons, where it is discharged by 7 pan feeders to a conveyor belt driven by a Dings magnetic pulley. This conveyor delivers the ore to a $\frac{3}{4}$-inch vibrating screen, the oversize from which goes to a Traylor gyratory crusher set to break to $\frac{3}{4}$-inch size. The discharge from this crusher is carried by belt conveyors to the fine-ore bins, where it joins the undersize of the vibrating screens. The fine-ore bin has a capacity of 400 tons and is discharged by 2 Hardinge belt-weight feeders to two 8-foot by 48-inch Hardinge ball mills in closed circuit with a 30-inch Akins, submerged-type, double-spiral classifier; the overflow from the classifier (minus 80-mesh) goes to a conditioner and thence to 12 Denver sub-A flotation machines connected to handle the tailings in series.

The feed enters the second cell, the concentrates from which, together with those from cells 3, 4, and 5, are fed to cell 1, which acts as a cleaner. Concentrates from cells 6, 7, and 8 are returned to cell 3, and concentrates from cells 8 to 12, inclusive, go back to cell 6 or 7. The tailings from cell 12 are cleaned in a 6-cell Shimmin air machine, the concentrates from which are returned to the ball-mill circuit. The tailings from the Shimmin cells are discharged through an 8-inch wooden pipe line to the South Fork of the Coeur d'Alene River about 2 miles from the mine.

The flotation concentrates containing about 55 percent of water are dewatered on a 6- by 10-foot Oliver filter, which reduces their moisture content to about 6 percent. The flotation reagents employed are lime, Air Float, and Barrett No. 4.

Figure 4.—Plan of the Sunshine mine.

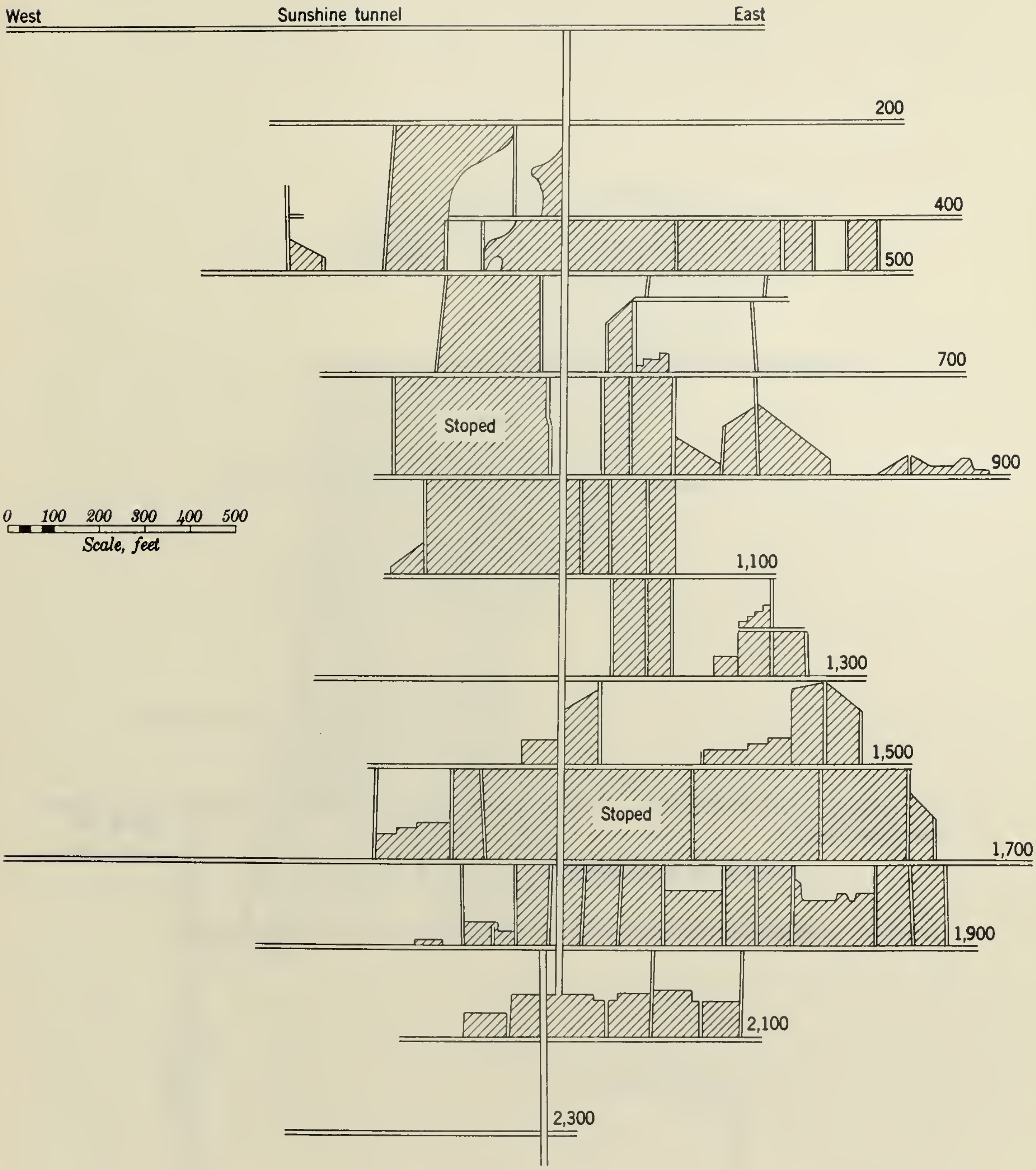

Figure 5.—Vertical longitudinal section of the Sunshine mine.

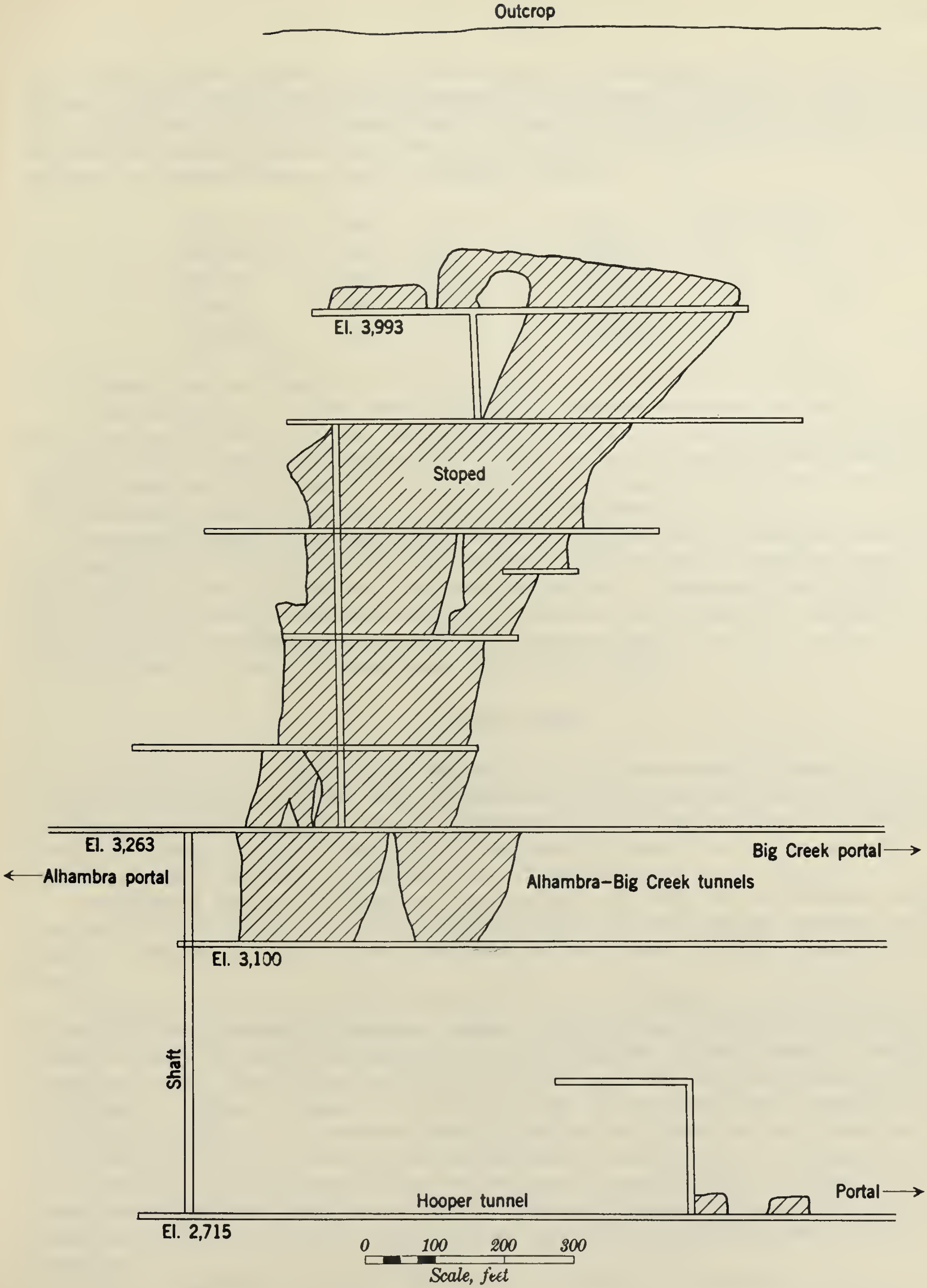

Figure 6.—Vertical longitudinal section of main workings of Crescent mine.

Present milling capacity is 475 tons per day, but adjustments now in progress will increase it about 50 tons and a later increase to 1,000 tons is planned to follow completion of the new shaft during the coming year. Average mill heads contain 45 ounces of silver per ton and average concentrates, 925 ounces per ton. The recovery of silver exceeds 97 percent. An average assay of the concentrates follows:

Gold	ounces	1.03	Iron	percent	29
Silver	do.	925	Sulphur	do.	38
Copper	percent	12.00	Arsenic	do.	1.60
Lead	do.	2.50	Bismuth	do.	.13
Antimony	do.	9.50	Insoluble	do.	3

Concentrates are shipped to the Bunker Hill & Sullivan smelter at Kellogg.

Costs and Wages.- In July 1935 mining costs averaged $3.59 per ton and milling costs $0.55, a total of $4.14 per ton. Other costs, exclusive of taxes and overhead, were $0.88 a ton, making a total cost chargeable to direct mining and milling operations of $5.02; of this amount $3.18 was paid for labor and $1.84 for power and materials. Total operating costs amounted to 13.79 cents per ounce of silver recovered.

The total number of men employed in November 1935 was 390, of which 270 were employed underground, 25 in the mill, 70 on construction, and 25 on staff and office work. Two 8-hour shifts are worked underground and three in the mill. Wages paid are: $5.25 for muckers; $5.75 for miners, increased to $6.80 in raises and shaft; $6.00 for pipemen and timber framers; $6.25 for timbermen, motormen, and repairmen; $7.40 for shift bosses; and $8.00 for carpenters. In the mill wages range from $5.50 to $6.50.

Crescent Mine

The Crescent mine, owned by the Bunker Hill & Sullivan Mining & Concentrating Co., is on the west side of Big Creek about 1½ miles above its confluence with the South Fork of the Coeur d'Alene River and a little downstream from the Sunshine mine. It includes three groups of claims, formerly known as the Big Creek, Crescent, and Alhambra, which extend from Big Creek to Elk Creek on the west side of the ridge.

Development work on the Alhambra group was begun in 1893, and the Crescent began production in 1924. The properties as now consolidated are credited with an output of approximately 3,000,000 ounces of silver, nearly all of which came from one ore shoot in the Crescent.

Mine.- The ore occurs along the hanging wall of the Alhambra fault which has been explored by three tunnels-the Alhambra, Hooper, and Big Creek. The portal of the Alhambra tunnel is near the head of the West Fork of Elk Creek about 2 miles east of Kellogg. The tunnel runs south about 3,000 feet to the Alhambra fault and then east along the fault for 9,000 feet. The Hooper tunnel is on the west side of Big Creek with its portal at an elevation of 2,700 feet. It runs west about 5,000 feet to a point 550 feet below the face of the Alhambra tunnel, with which it is connected by a vertical shaft. The portal of the Big Creek tunnel is slightly north of that of the Hooper tunnel but 550 feet above it. The Big Creek tunnel runs west almost 3,000 feet to intersect the main workings of the mine at the level of the Alhambra tunnel. Figure 6 shows a vertical longitudinal section of the main workings.

The ore consists principally of quartz and siderite associated with argentiferous tetrahedrite, galena, and pyrite. It averages about 25 ounces of silver per ton. In the upper workings where the ore is oxidized small quantities of cerussite and well-crystallized native silver are found.

The Alhambra fault, along which the ore occurs, is a steep reverse fault, dipping south about 50° near the surface but 65 to 83° at the level of the Alhambra tunnel. The hanging wall is of Revett quartzite, while the foot wall is of the characteristic thin-bedded Wallace formation. The ore is always separated from the foot wall by a few inches of dark-colored gouge and ranges in width from 1 inch to 12 feet. As elsewhere in the district, the mineralization occurred subsequent to the faulting.

In the Alhambra tunnel the mineralization was spotty all along the wall until a good ore shoot was found in Crescent ground 10,500 feet from the portal. The shoot on this level is 300 feet long and has been mined for 800 feet above the level and 150 feet below it. At the top the shoot was 800 feet long. There are 400 feet of unmined backs on this shoot above the level of the Hooper tunnel.

Mill.- The Crescent mill is on Big Creek just below the portal of the Hooper tunnel. It has a capacity of 120 tons per day, but in November 1935 it was operating only one 8-hour shift and treating 40 tons.

It receives ore from the Big Creek tunnel by way of a bucket tramway and ore from the Hooper tunnel by electric haulage, the mine cars discharging directly to the coarse-ore bin. From this bin the ore goes to a 9- by 15-inch jaw crusher and thence to 30- by 12-inch rolls set to crush to 3/8-inch size. It is next ground in a 6- by 4-foot Hardinge mill until 60 percent of the discharge is minus 200-mesh size. This discharge goes directly to a Denver Equipment Co. unit flotation cell, which makes a high-grade concentrate accounting for about 45 percent of the total silver recovered. The tailings from the cell go to a Dorr classifier, the sands from which go to a ball mill and the overflow to thickening tanks. The clear overflow from the tanks is wasted, and the underflow containing about 28 percent of solids goes to a Denver A 10-cell Fahrenwald flotation unit. The concentrates from the first 3 cells of this unit go to a pneumatic cleaner, and those from the last 7 cells are returned to cell 1. The tailings from the last 7 cells are wasted. The moisture content of the concentrates going to the Oliver filter is reduced from about 50 to approximately 6 percent. The tailings from the cleaner cell are in closed circuit with the ball mill.

An average analysis of the concentrates follows:

Silver........	ounces per ton	250	Arsenic..........	percent	4.5
Iron............	percent	29	Antimony........	do.	4
Sulphur......	do.	32	Copper............	do.	8
Lead............	do.	6	Zinc................	do.	0
Bismuth......	do.	.002	Insoluble......	do.	4

Average mill heads contain 25 ounces of silver per ton. Milling costs average 85 cents per ton although the mill is operated at only one third capacity.

Tailings are discharged through an 8-inch wooden pipe line 2 miles long to the South Fork of the Coeur d'Alene River.

The reagents used in flotation are lime, Barrett No. 4, pine oil, and xanthate (Z-3); the last is employed as a promoter.

The sulphide ores from the Crescent are well-adapted to flotation; a recovery of 95 percent of the silver is made when sulphides alone are treated, but with mixed sulphides and oxides the recovery ranges from 80 to 85 percent. As the recovery in millimg oxidized ores is relatively poor, high-grade ore from the oxidized zone, averaging 100 ounces of silver per ton, is sorted out in the stopes and shipped directly to the Bunker Hill & Sullivan smelter.

About 60 men are employed in the mine and 2 in the mill.

Sunshine Consolidated

The Sunshine Consolidated property, including about 1,350 acres in process of patenting, is on the west side of Big Creek opposite and adjoining the Sunshine mine. It contains the Sunshine shear zone and presents the same geological conditions as the Sunshine. The Crescent mine bounds it on the northwest, and other Bunker Hill & Sullivan ground lies to the west.

Diamond drilling is said to have disclosed recently 2 veins in the shear zone, and a contract has been made with the Sunshine Mining Co. to drive 300 feet into the Consolidated ground from its west 1,700-foot level. This exploratory drift should be in Consolidated ground by February.

The Sunshine Consolidated has contracted with the Sunshine Mining Co. for hoisting service through the new four-compartment shaft of the Sunshine being sunk just across Big Creek in Sunshine ground.

Globe

The Globe is on the west side of Big Creek about a mile above the Sunshine mine and immediately south of the Sunshine Consolidated; it is less than 3 miles south of the main highway (U. S. No. 10). A 460-foot tunnel runs a little north of west toward the Big Creek fault.

The property recently has been equipped with a bunkhouse, an eatinghouse, a changeroom and a blacksmith shop, and a room has been prepared for the installation of a 500-cubic-foot compressor. The management expects to employ eight men for driving the tunnel after the compressor is installed.

Metropolitan

The Metropolitan property is 1¼ miles above the Sunshine mine on the east bank of Big Creek.

Development consists of a main tunnel running northeast into the hill for about 1,200 feet, with a crosscut at the 900-foot point that runs east for 1,800 feet and then south for an additional 2,200 feet. Almost opposite this crosscut is another about 100 feet long to the northwest, at the end of which is an inclined shaft 230 feet deep. It follows a vein 1 to 20 inches thick, dipping 55°.

The ore minerals are pyrite and some galena in quartz with a little tetrahedrite.

Six men are employed at this mine.

Coeur d'Alene Big Creek

On the west side of Big Creek the Coeur d'Alene Big Creek property lies south of the Big Creek group of the Crescent mine and north of the Sunshine Consolidated. On the east side of the creek it extends along the northern boundary of the Sunshine mine.

It is an old property; a tunnel driven west about 1,000 feet from near the creek level is probably in the footwall of the Sunshine shear zone. No work is being done upon the property at present.

Silver Syndicate

The Silver Syndicate property, formerly the Little Sunshine, belongs to the Silver Syndicate, a group in process of incorporation. It is east of Big Creek and north of the Coeur d'Alene Big Creek, the Sunshine mine, and the Polaris mine.

Approximately 8,000 feet of drifting was done on this property, chiefly on a tunnel level starting on the east side of Big Creek. The tunnel is said to have intersected the Polaris fault, in which a raise was driven 300 feet. It is reported that the raise showed a 1½-foot width of milling ore carrying 18 ounces of silver a ton.

A shaft was sunk 500 feet from the tunnel and 1,200 feet of drifting at the 400-foot level is said to have shown 2 to 18 inches of milling silver ore and a small body of zinc ore.

Big Creek Silver, Inc.

Big Creek Silver,Inc., a new company, has acquired the old Elgin & Ogden property on the east side of Big Creek north of the Silver Syndicate property. Little work has been done in this ground except that a short prospecting tunnel has been driven, but a small crew is now working there.

Polaris and Spring Gulches

The waters of Polaris and Spring Gulches flow into the South Fork within a mile east of Big Creek. Both are less than 2 miles long and very steep. They head in the main ridge in the vicinity of the Polaris fault and the Sunshine shear zone. The Polaris, Chester, Mineral Mountain, and Purim properties lie within the drainage basins of these gulches.

Polaris Mine

The Polaris mine is at the head of Polaris Gulch, a little more than a mile from the valley of the South Fork, from which it is reached by a steep, winding road several miles long. It is about a mile east of Big Creek on the east side of a mountain ridge on whose western slope the Sunshine mine is situated. This was one of the first mines worked in the district; it was being operated at the time of the epoch-making Bunker Hill discovery in 1885. The value of its past production is said to have exceeded the $250,000 of record.

Old workings near the crest of the ridge are in the Wallace formation, which gives place to the St. Regis a short distance to the north at the Polaris fault, named after this mine.

Recently an old tunnel at a lower elevation, the Polaris, has been extended along the vein. Some narrow stringers of silver ore were encountered in it. From this tunnel a winze was sunk 920 feet, where a crosscut to the south disclosed a silver vein several feet wide.

Toward the winze bottom another tunnel about 2 miles long, the Silver Summit, is being extended by the Polaris Co. It starts at the mouth of Rosebud Gulch at approximately the same elevation as the main-adit level of the Sunshine mine.

Connection of the Silver Summit tunnel with the winze will afford an excellent opportunity for exploration of Polaris ground laterally and at depth.

The Polaris Co. is controlled by the Hecla Co., and the Newmont Mining Co. is associated with the Hecla in the enterprise.

Chester Mine

The old Chester mine adjoins the Polaris mine on the east. The two are much alike; they are traversed by the Polaris fault, and the Chester, as well as the Polaris, is now controlled by the Hecla Co. The Silver Summit tunnel will provide access to both properties.

Two tunnels were driven into the Chester long ago on a westerly course from Spring Gulch, one of them being over 1,000 feet long. Because of caving neither is now accessible. Some shipments of silver ore are said to have been made.

A drift from the Silver Dollar tunnel was also extended into the Chester from its eastern boundary.

Mineral Mountain

Mineral Mountain, a new property in Spring Gulch, is north of the Chester. A tunnel has been driven west about 150 feet from the gulch as an exploratory crosscut.

Purim Group

The Purim group of claims is an area east of the Sunshine and south of the Chester, through which the Sunshine shear zone almost certainly extends, although there are no workings actually exposing it. The shear zone is seen, however, in the Silver Summit tunnel east of the Purim.

A half interest in the Purim group is owned by the Silver Dollar, and the other half by individuals.

Rosebud Gulch

Rosebud Gulch 2 miles east of Big Creek is of great importance as a natural portal for access from the east to the properties at the western end of the silver belt. From its junction with the valley of the South Fork two long tunnels have been driven south, the Silver Dollar from a little west of the gulch and the Silver Summit from its east side.

Silver Dollar

The Silver Dollar property has, in addition to half interest in the Purim group, another large area north of it that surrounds the east end of the Chester. This part of the property is held by long-term lease from the old Lincoln Co.

The Silver Dollar tunnel starts in the valley of the South Fork a short distance west of the mouth of Rosebud Gulch. It has been driven in a southerly direction more than a mile. About 4,000 feet in from the portal it cuts the Polaris fault, upon which are drifts both west and east totaling about 1,500 feet.

Silver Summit

The Silver Summit property includes a large area of ground east of the Silver Dollar and the Purim group. It extends south from the mouth of Rosebud Gulch through the Polaris fault and Sunshine shear zone to within half a mile of Big Creek fault.

A tunnel in this property will be extended to the bottom of the Polaris shaft (see Polaris Mine). It is driven due south almost a mile and provides a valuable exposure of the rock cross-section from the valley through the Polaris fault and Sunshine shear zone.

The Polaris fault is cut 2,100 feet from the portal. The north side of the Sunshine shear zone is 2,100 feet farther in at the 4,200-foot point. Although no ore is exposed there is pronounced mineralization in considerable widths. The shearing is several hundred feet wide.

At the 4,200-foot point a drift to the west has been carried 1,000 feet or more, and it is now being extended as a crosscut toward the Polaris shaft. The total length of the tunnel from its portal to the connection at the shaft will exceed 11,000 feet.

Nellie Mine

The Nellie mine of the Empire Silver Corporation is on the Polaris fault on the east side of Rosebud Gulch. It was once a fairly active operation said to have shipped silver ore valued at $100,000; it is now inactive.

McFarren Gulch

In McFarren Gulch, which joins the valley of the South Fork opposite the town of Osburn about 3 miles east of Big Creek, are 5 properties--the Plain View, Mineral Point, Merger Mines, Idaho-Montana, and Silver Standard.

Plain View

This property lies in an extension of the Nellie to the east as far as the west side of McFarren Gulch. It is crossed by the Polaris fault. There are some old workings, but the property is not active at present.

Mineral Point Mine

The Mineral Point mine of the Coeur d'Alene Mines Corporation is on the east side of McFarren Gulch about a mile south of the valley of the South Fork. It is reported to have been a former producer of high-grade ore valued at nearly $100,000.

The old workings comprise several tunnels near the top of a mountain on narrow veins north and east of the Polaris fault. They strike east and west, dip 45° south, and cut bedding of the Wallace formation at a slight angle. Mining was rendered difficult and finally terminated by faulting.

Work recently has been resumed, and a tunnel is being driven from near the bottom of the gulch 1,000 feet below the crest of the hill. This tunnel was started south of the Polaris fault. It encountered a strong vein containing quartz and siderite, although no important ore shoot has yet been found in it. Typical silver-belt ore containing silver in tetrahedrite with siderite has been found recently in the 200-foot-level tunnel.

Fifteen to twenty men are employed.

Merger Mines

The property of Merger Mines Corporation lies west of the Mineral Point mine and south of Plain View. As it extends far south it probably is crossed by the Sunshine shear zone. Two upper tunnels show vein material, and a lower tunnel is being driven from near the bottom of the gulch. Drilling from it is now being conducted and is said to have encountered the large vein discovered in the 1,000-level tunnel of the Mineral Point mine. About 10 men are employed.

Idaho-Montana

The Idaho-Montana property is south of the Mineral Point mine and east of Merger Mines. An old tunnel about 1,300 feet long exposes considerable shearage. Recently some showing of typical silver-belt ore was reported to have been found.

Silver Standard

The Silver Standard is south of the Idaho-Montana and east of Merger Mines. As yet only a short tunnel has been driven.

East of McFarren Gulch

Mining developments inspired by the success of the Sunshine mine extend only about 4 miles southeast of Big Creek to the eastern limit of the McFarren Gulch drainage basin. East of that area no properties are being operated at present, but a considerable production of silver ore was made from the old Argentine mine in Argentine Gulch, and some showings of similar ore were found east of it as far as Placer Creek.

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INFORMATION CIRCULAR

PROGRESS REPORT ON INVESTIGATION OF DETACHABLE ROCK-DRILL BITS

BY

MCHENRY MOSIER

I.C. 6877
February 1936.

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PROGRESS REPORT ON INVESTIGATION OF DETACHABLE ROCK-DRILL BITS[1]

By McHenry Mosier[2]

INTRODUCTION

The object of this investigation is to determine the field for the use of detachable bits in metal mines of the United States by correlating the data developed through the experience of mines using these bits. Most of the metal-mining districts in the Middle Western and Eastern States already have been visited.

The purpose of the present circular is to define the problem and to discuss the factors that must be considered in order to determine, under different operating conditions, the advantages and disadvantages of this type of bit. Some of the data obtained to date are presented to emphasize points made in the discussion. It is hoped that, as the study progresses, a great many data on drilling will become available. It is evident that since in some cases these data will be confidential they cannot, therefore, be published except as they may be included in average figures covering several different mines. Some data already secured are of this nature. Although considerable information on drilling operations has been gathered at different mines, there were only a few instances in which there were available complete data on all the factors entering into the total cost of drilling with detachable bits and with conventional bits on regular drill steel.

In view of the interest in this subject among metal miners, the Bureau of Mines has completed arrangements whereby mine operators who so desire may avail themselves of the services of one of its engineers, without cost to them, in the making of time studies and assembling of cost data required to obtain a complete picture of the comparative merits of detachable bits and of conventional bits on regular drill steel under the particular conditions prevailing at their individual mines.

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6877."

2/ Senior mining engineer, Mining Division, U.S. Bureau of Mines.

ACKNOWLEDGMENTS

Most of the information upon which this report is based was derived from the experience at the mines visited and was made available through the courtesy and cooperation of the officials at these operations.

HISTORY

As early as 1865 a patent was granted in the United States for a three-bladed hand-drill bit. This was a detachable bit that was screwed on the drill steel. Since that date a large number of patents have been issued for various types of detachable bits on both hand and machine drill steel.

The use of detachable bits, which is continually expanding, is no longer confined to small mines and contractors. Although such bits are now manufactured from high-carbon steel, which has greatly improved their quality, yet they must be considered as being in the development stage still. One possible method of further increasing their present degree of excellence is through the use of special alloyed steel. In fact, recent experience on the Rand in South Africa with tappet machine drills having hot-milled conventional bits made from chrome-molybdenum drill steel shows a reduction in the cost of drilling. R. S. G. Stokes states3/:

> The eternal quest for a detachable bit has not been wholly abandoned, but, with the great reduction in number of borers used, the objective has lost much of its appeal; and, with the use of lighter steel for quicker drilling, a serviceable bit and stem have become more difficult to design. Experiments are in progress with a detachable, resharpenable bit, in which the cushioning member in the stem socket for the tapered bit is a lining of soft metal.

BASIC PRINCIPLE OF DETACHABLE BIT

The basic principle of the detachable bit is the construction of a drill steel shank and a bit in two separate units. This permits (1) the bit to be made of a steel that can be hardened to a high degree; (2) the shank or rod to which the bit is attached to be made of a different steel, selected for toughness and resistance to fatigue; and (3) the bit, on account of its small weight and bulk, to be conveniently and economically transported between drill shop and working face.

TYPES OF DETACHABLE BITS

At present six major types of detachable bits are produced in quantity for use in metal mines in the United States. Some of these have several patterns. The first was developed in 1918, while the most recent was produced in 1934. All are illustrated in figure 1. Several other types are now in process of experimental development at various properties. In type 1

3/ Recent Developments in Mining Practice on the Witwatersrand: Bull. Mining and Metallurgy, London, November 1935.

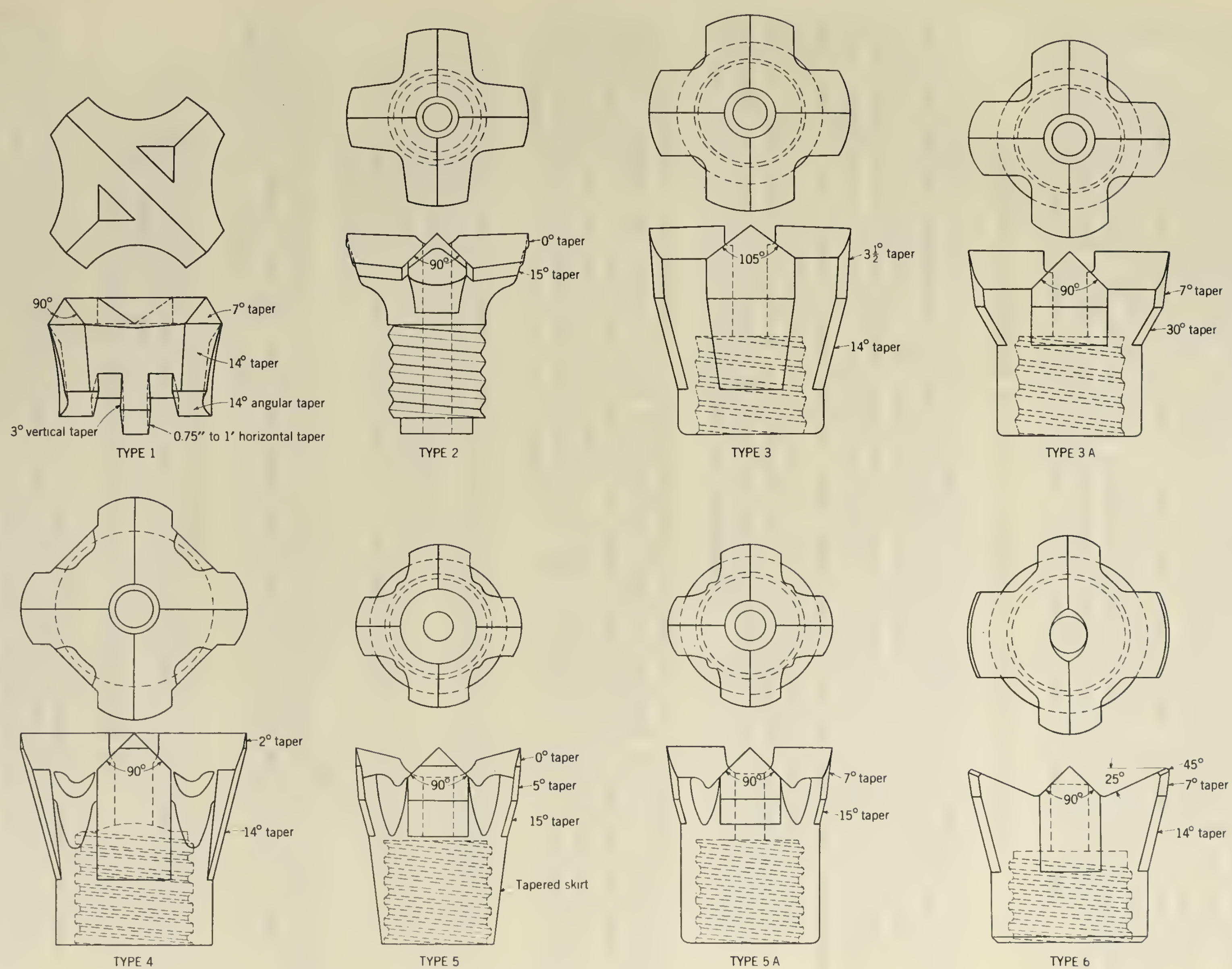

Figure 1.–Some common types of detachable rock-drill bits.

the bit is attached to the shank or rod by means of a tongue-and-groove joint; in type 2 a threaded coupling is used, with the base of the bit bearing on the end of the rod; in type 3 attachment is accomplished by means of a reverse buttress screw thread with a recessed face in the bit contacting the end of the rod; in type 4 a modified acme screw thread is used, with the skirt of bit bearing against a shoulder on the rod; in types 5 and 6 a modified acme screw thread with a recessed face in the bit bearing against the end of the rod is the method employed. Some distinctive designs of detachable bits are shown in the figure illustrating the several types.

FACTORS CONSIDERED IN THE COMPARISON OF DRILLING COSTS

In order to decide whether detachable bits would be advantageous to any particular operation in which regular drill steel with conventional bits is used, it is necessary to have accurate detailed information as to the total cost of all the items involved in each system of drilling. To get a true comparison of their respective merits, it is then essential to reduce the cost of all these items to the cost per foot of hole drilled, for each kind of bit. In preparing these costs, some mines have found that some factor previously regarded as unimportant in its relative effect was, in reality, the critical feature that definitely established the difference in the cost of drilling with the two methods.

From data collected at a considerable number of metal mines, it is possible to point out some of the conditions most favorable to the introduction of detachable bits and those conditions most favorable to the use of conventional bits on regular drill steel. In general, the operating conditions at a mine using regular drill steel that may be said to favor the adoption of detachable bits are the following:

1. High transportation and distribution costs for drill steel.
2. Large drill-steel loss.
3. High drill-sharpening shop costs.
4. Low footage drilled per bit.
5. Inability of the mine shop to produce a bit of uniformly high quality.

Although detachable bits can drill any type of ground, yet, from a cost standpoint, those conditions that are unfavorable to the adoption of detachable bits at their present cost are:

1. The reverse of those listed above.
2. Exceedingly abrasive ground, with consequent rapid loss of the gauge of the bit.
3. Any condition that destroys the bit in a single run.

The only way in which the effect upon drilling costs of each of the various factors entering into the total cost can be evaluated is by means of data developed through actual drilling tests conducted at the particular mine under consideration. At some properties, conditions favoring the use of detachable bits may not prevail generally throughout the mine but may

be limited to certain classes of work. At several operations of this kind, regular drill steel is used for ordinary routine work and detachable bits for special jobs. The latter comprise such tasks as long-hole drilling by means of jointed rods and the drilling of headings that are particularly inaccessible and faces that cannot be drilled satisfactorily with conventional bits forged on regular drill steel at a local shop.

The tangible factors that affect the total cost per foot drilled with detachable bits and with conventional bits on regular drill steel are enumerated below:

1. Cost of transporting drill steel and bits from sharpening shop to working face and return.
2. Shop operating expense for reconditioning drill steel and bits.
3. Cutting speed or rate of penetration.
4. Number of feet drilled per bit dulled.
5. Feet drilled per machine shift.
6. Loss of drill steel from all causes.
7. Drill machine repairs.
8. Hazards in transportation and use of drill steel and bits.
9. Capital expense.

DISCUSSION OF FACTORS

The following discussion of these factors presents some preliminary operating data for detachable bits and for conventional bits on regular drill steel, and shows their relation to the cost per foot of hole drilled.

1. Cost of transporting drill steel and bits from sharpening shop to working face and return. The cost of transporting drill steel, with its corresponding reflection in the cost of mining, is often the most difficult to determine of all the drilling expenses for any given mine. Yet, by means of time studies covering a long enough period, even this elusive cost can be determined with a reasonable degree of accuracy. Included in cost of transportation is the expense of delivering steel or bits from the shop to the collar of the shaft, of handling them through the shaft, and of distributing them on the mine level or through manways and raises to the working face, as well as the expense of returning them to the shop. When mines are operating at a reduced rate of output, the true cost of transporting the steel may not be apparent, but when operations approach the maximum rate of production this cost becomes more obvious. In those mines where supplies are handled through an ore-hoisting compartment of a shaft, the loss of effective hoisting time will become more serious as maximum capacity is approached. At most mines the usual procedure in segregating costs is to charge transportation with its proportionate share of hoisting expense, based on the number of trips made through the shaft and the time involved. In mines where tool nippers are employed underground, the time they spend in handling steel is, of course, included in the transportation charge. When the miners themselves transport the drill steel, the question arises as to whether this transportation is a debit against the cost of drilling. In the case of miners

drilling in open stopes, particularly with sublevels, and in other similar instances, where there always are plenty of faces to drill, the more time the miners lose in handling steel the less drilling can be done. When miners are employed on development work with only one working face available, it may be more difficult to utilize effectively the time saved in handling drill steel through the adoption of the detachable bit. Even in such an instance, there is frequently enough related work in the heading itself to occupy the miners fully after the round is drilled, so it would appear that the actual time spent by the miners in transporting drill steel is a definite and proper charge against the cost of drilling.

2. Shop-operating expense for reconditioning drill steel and bits. In the shop-operating expense, in addition to labor and supplies, compressed air for operating drill-sharpening machines should be measured and charged for at the average unit cost for air compression. There are also included in this shop expense such items as fuel, electricity, water, tools, and repairs to equipment. Thus, the total cost per bit sharpened can be found and then converted into cost per foot drilled.

At some mines, detachable bits are reground two or more times and then discarded; at other mines, instead of discarding, retempering is introduced with subsequent additional regrinding; at one mine this latter practice is followed by reforging, retempering, and still further regrinding. On the present market, the original purchase price of a detachable bit plus the shop cost of reconditioning is normally greater on a "times-used" basis than the corresponding cost of only those particular items for regular drill steel. To determine whether there is a net saving through the use of detachable bits, all the other tangible factors must be considered.

3. Cutting speed or rate of penetration. With other conditions remaining constant, the design or pattern of the bit and its gage largely influence the cutting speed. Many things enter into the design of a bit. The first consideration is the general type, as, for example, chisel bit, cross bit, or rose bit. Then follow such details as clearance tapers, whether single or double, and the degree of each; the angle between faces, which may be acute, obtuse, or 90 degrees; the dimensions of the wings, including their heighth and width; the amount of reaming effect from the edges of the wings; and the position of the hole through the bit, which may have a center or side outlet. In general, the usual patterns and designs of bits in common use with regular drill steel are duplicated in detachable bits.

One of the principal points in design of bits is the amount of clearance for cuttings required for rapid drilling in a given formation. Small gages drill faster than large gages. One mine reports a 58-percent increase in drilling speed in granite for a 3/4-inch decrease in the gage of the bit, which, in this instance, is roughly inversely proportional to the area of the hole drilled. Another mine, using conventional bits on regular drill steel, has effected appreciable economy in the cost of mining by decreasing the size of the drill hole and increasing the strength of the explosive. The saving resulting from faster drilling with smaller-gage bits is far

greater than the increased cost of the higher-grade explosive required to give the same fragmentation of the ore. At this mine, the "starter" for a 10-foot hole with its 1-3/8-inch gage is smaller than the "finisher" used at many mines, and the drilling speed has been increased correspondingly.

The tendency among users of detachable bits is toward smaller gage changes. Most of these operators use a "following gage", in which the bits, instead of being ground down to a predetermined gage, are first sharpened and then sorted into sizes having, at some mines, as small a differential as 1/64 inch. As conventional bits on regular drill steel in normal practice cannot be made with such a small gage change, the advantage in this regard is with the detachable type. However, for very small gages, the detachable bit is at a comparative disadvantage. For the types now in commercial production in the United States it is not possible to construct a bit with as small a gage as can be forged on regular drill steel of corresponding cross section. Ordinarily, the smallest detachable bit is 1-1/4-inch gage, and this is used on a 7/8-inch rod.

The hardness and the quality of the cutting edge of a bit are factors that depend on the composition of the steel and the corresponding heat treatment.

One of the largest items in the cost of drilling is the labor cost per foot of hole for operating the drill. The cutting speed largely influences this cost, though not in direct proportion, because the drill is actually cutting during only a small proportion of the total lapsed time of any shift. With a constant air pressure, the compressed-air consumption per foot of hole depends in a great measure on this cutting speed. At 10 mines where comparative records were available the average cutting speed with detachable bits was 22 percent faster than with conventional bits on regular drill steel.

4. Number of feet drilled per bit dulled. The footage drilled per bit dulled has as much influence on the cost of drilling as the cost of the bit itself. It is the factor employed in converting the cost per bit into cost per foot of hole drilled. The number of feet drilled per bit dulled with conventional bits on regular drill steel depends, in no small measure, upon the degree of perfection attained in forging and hardening the bit. This, in turn, depends on the skill of the personnel in the local shop as well as on the sharpening and heating equipment and the means for heat control. This process requires specialists, but, at the average mine shop, it is not always possible either to procure or to retain the services of men properly trained in such metallurgical technique.

In the manufacture of the detachable bit all processes are under positive scientific control by skilled operators with the best equipment obtainable. The result is an absolutely uniform product capable of the maximum service possible for the carefully selected steel specified in its construction. The steel now commonly used for manufacturing detachable bits in the United States has a higher carbon content than regular drill steel, with consequent greater hardness and resistance to abrasion. These factors give a longer life to the bit.

Fourteen mines that have had extensive experience with detachable bits show an average increase in footage drilled per bit of 28 percent over that of the the conventional bit on regular drill steel.

5. Feet drilled per machine shift. The footage drilled per machine shift establishes the labor cost per foot of hole. Under circumstances that permit continuous drilling, the cutting speed becomes more important as a factor in the footage drilled. Any increase in cutting speed that might result from the introduction of the detachable bit is at once apparent in the total number of feet drilled. Of course, any delays in drilling due to shortage of sharp steel will increase the labor cost per foot of hole. In such a case, particularly with regular drill steel, miners may use extremely dull bits, which slow the drilling speed and increase machine-drill repairs. On the other hand, when only the required footage for a given round can be drilled for lack of sufficient working faces to keep the machine drill employed during the entire shift, the fixed footage is, in fact, the maximum attainable, regardless of increased cutting speed. Even in this case the shorter drilling period is advantageous, as it reduces the compressed-air consumption and also permits the miner to do other related work.

6. Loss of drill steel from all causes. The loss of drill steel includes wastage in the process of heating and forging the bit, abrasion in actual drilling operations, breakage of drill steel, whether from normal fatigue or from improper handling in the shop and in transportation, and the disappearance of unaccounted-for pieces.

Abrasion of bits in actual drilling is reflected by the loss in gage. It is less with detachable bits than with regular drill steel at the mines visited. At one property, which kept precise records of drilling tests, the detachable bits showed only one-sixth as much abrasion per foot of drilling as the conventional bits on regular drill steel.

The miner who uses detachable bits has so few rods in service that he finds it much easier to avoid their unaccounted-for loss than with the larger required number of pieces of regular drill steel.

On a fixed scale of operations, the total loss of drill steel is measured accurately by the normal average monthly or annual amount of replacement steel required to keep a constant stock in service. For regular drill steel it may be as low as 0.007 or as high as 0.770 pound per foot of hole drilled. At four of the mines visited there was available a comparative record of the loss of drill steel for the two classes of bits. The average reduction in this loss through the adoption of detachable bits at these four mines was 48 percent.

7. Drill machine repairs. A correct comparison of the cost of drill-machine repairs before and after the introduction of detachable bits cannot be made until a long enough time has elapsed following the change to eliminate entirely the influence of the use of regular drill steel. Thus, under the new conditions, it might be a year before the normal cost would be evident.

To get a true comparison of these costs, all other factors, such as the formation drilled, lubrication of the machine drills, and general mine-operating conditions, must remain constant. No reliable comparative figures for drill-machine repairs are available as yet at the mines visited, although several properties that have changed to detachable bits show a tendency toward a smaller drill-machine repair cost. The most plausible explanation for this is the suggestion that the detachable bits usually are not run to as dull a cutting edge as are regular bits, and therefore do not impose as severe shock on the drill machine.

8. Hazards in transportation and use of drill steel and bits. It is sometimes difficult to eliminate all the hazards involved in transporting many comparatively heavy pieces of regular drill steel of variable lengths. From a safety viewpoint a rather hazardous condition is brought about when these pieces are handled in and out of a skip or cage and again on the mine levels under live trolley wires. The use of detachable bits reduces these hazards because fewer rods are required.

It is customary to strike the detachable bit lightly in order to loosen it. By using a small hammer of mild steel for this purpose, the danger from flying splinters of steel is eliminated.

9. Capital expense. This includes the cost of construction of the drill-sharpening shop building, the cost of the shop equipment and of its installation, together with the cost of the initial stock of drill steel and bits.

For a large operation having many bits to sharpen every day, less equipment is necessary for reconditioning detachable bits than for conventional bits on regular drill steel, and there are fewer pieces of long drill steel to handle. Consequently, a smaller shop building and a lower construction cost is required. But, for a smaller operation the reverse is true; because, after adopting detachable bits it is usually found convenient to have a drill-sharpening machine for shanking and other work; and, in addition, it is necessary to install apparatus for grinding bits. If detachable bits are rehardened, muffle furnaces and especially designed quenching arrangements will be necessary for the best results in this work. The kind of equipment required for preparing one end of the rod to receive the bit depends upon the particular method of attachment used. The reconditioning of bits and rods at a very small operation without shop facilities sometimes can be done at some other plant with suitable equipment for such work. Thus further capital expense can be avoided, though probably at an increased unit cost for reconditioning bits and rods.

A smaller stock of drill steel is required for making detachable bits than for regular bits. At one mine only one-sixth the amount of drill steel formerly used was required after the use of detachable bits was adopted. The more remote and inaccessible the location of the mine, the more important this item of inventory becomes, because of the high delivery charges for drill steel.

The cost of making up the initial stock or sets of regular drill steel and the rods for detachable bits is sometimes included in this inventory cost under capital expense.

The capital expense can be translated into cost per foot drilled by adding the depreciation to the annual interest on the investment and finding the ratio of this sum to the annual footage drilled.

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NOTES ON TESTING THE EXPLOSIBILITY OF COAL DUSTS AND A PROPOSAL TO HAVE AN INTERNATIONAL TEST METHOD

BY

George S. Rice and H. P. Greenwald

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INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

NOTES ON TESTING THE EXPLOSIBILITY OF COAL DUSTS AND A PROPOSAL TO HAVE AN INTERNATIONAL TEST METHOD[1]

By George S. Rice[2] and H. P. Greenwald[3]

The following paper was presented at an International Conference on Mine Safety Research held at Dortmund, Germany, September 23-28, 1935, inclusive. This conference was attended by representatives of Belgium, France, Germany, Great Britain, and the United States and, by invitation, Czechoslovakia, Poland, Spain, and Russia.

The first international conference of representatives of mine-safety research stations was held at the United States Bureau of Mines Pittsburgh Experiment Station in 1912.[4]

In 1924, the Safety in Mines Research Board of Great Britain entered into cooperation with the United States Bureau of Mines for interchange of research workers, of materials, of instruments for standardized testing, and of progress reports on problems of mutual interest. In 1929, the Safety in Mines Research Board undertook similar though more limited cooperative agreements with the French experiment station at Montlucon and the Belgian station at Frameries.

The favorable results of these cooperative arrangements led the Board to invite officials in charge of mining research at experimental stations in Belgium, France, Germany, the United States, Czechoslovakia, and Poland to attend a conference at Buxton, England, in July 1931.[5] Representatives of the last two countries were unable to attend. At this conference, a further agreement was made and subsequently officially approved by the countries represented for an international interchange of confidential quarterly reports and of special reports on research in mine safety and health. This interchange still continues.

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6878".

2/ Chief mining engineer, U. S. Bureau of Mines.

3/ Senior physicist, U. S. Bureau of Mines.

4/ Rice, G. S., International Conference of Mine Experiment Stations, Pittsburgh, Pa., September 14-21, 1912: Bull. 82, Bureau of Mines, 1914, 99 pp.

5/ Safety in Mines Research Board of Great Britain, International Conference on Safety in Mines at Buxton, 1931: Paper 74, 1932, 67 pp.
Rice, G. S., International Conference on Mine-Safety Research at Buxton, England, July 1931. Inf. Circ. 6670, Bureau of Mines, 1932, 19 pp.

The third international conference of this kind was held at Montlucon, France, in 1933, but the United States was unable to send a representative at that time. The fourth conference was held at Dortmund as mentioned, and the present paper was presented there by the senior author. It proposes standardization of methods of testing the relative inflammability of coal dusts, beginning on a laboratory scale. This proposal was unanimously approved. The papers and discussions of the Dortmund meeting probably will be published in the German language only, and it seems advisable that the Bureau of Mines publish them in English with such minor changes as may be made necessary by the manner of publication.

NOTES ON TESTING THE EXPLOSIBILITY OF COAL DUSTS

Before considering questions of standardization, the authors of this paper examine some of the underlying principles of large-scale testing of the explosibility of coal dust and their relation to conditions in commercial mines. The term "explosibility" is used here with the meaning attached to it by investigators of the United States Bureau of Mines in the field and in large-scale test work and is equivalent to the British term "inflammability", which, in Bureau of Mines work, is reserved for use in tests in which combustion of the coal dust does not extend beyond the sphere of influence of the source of ignition. Laboratory tests fall in this category.

INITIATION OF A COAL-DUST EXPLOSION

By definition, this phase of a coal-dust explosion covers the period during and the area in which the source of ignition is active. Whether or not the incipient explosion may be self-sustaining outside these limits is discussed later. Coal dust being present initially, either naturally in a mine or by provision of the investigator for his test, two things are needed to initiate an explosion - first, a means of dispersing the dust as a cloud in air and, second, a means of igniting the dust cloud. These may be condidered separately.

Formation of a Coal-Dust Cloud

A concussion or disturbance of some kind is needed to raise quiescent dust and disperse it in air. In commercial mines, this dispersion can occur either with or without a source of ignition of the cloud being present; details need not be presented to an assembly familiar with the numerous ways in which this can happen. Although the Bureau of Mines also experiments with electric-arc ignition of coal dust raised in a dense cloud by a blast of air, this paper will deal only with the case in which the source both raises and ignites the dust cloud, as by blow-out shots of explosives or by gas explosions. This is the system used in all extended series of large-scale tests of which the authors have knowledge.

Energy must be expended to form a cloud from quiescent dust, and the quantity of energy required will vary with the specific gravity of the dust, size of the particles, condition of the surface of the particles insofar as

dispersion is affected by such things as electrostatic charges and moisture films, relative amount of inert dust present, if any, location of the dust in the cross-section of the passageway in which the cloud is formed, and whether all the dust present or only a part thereof is dispersed. This energy must be supplied by the source of ignition. The dust-raising power of sources of ignition used by different investigators has varied widely in accordance with their respective ideas. Three sources (other than the one not yet standardized, namely, electric ignition of a premade dust cloud) have been used at the Experimental mine, differing principally in their ability to form a dust cloud.[6] The two factors, ability of the dust to be dispersed and power of the source to disperse it, are fundamental considerations in initiation of a coal-dust explosion, whether in a commercial mine or in a testing gallery.

Ignition of a Coal-Dust Cloud

Heat must be supplied by the source of ignition to bring the coal dust to its ignition temperature. Not all the heat available for this purpose is used thus. There are direct losses to the surrounding walls through heating of inert or incombustible particles in the dust cloud (such as rock dust) and through heating of coal dust present in excess of the amount that can be burned by the quantity of air present; the conditions of this last case preclude liberation of energy through combustion to balance that absorbed. Vaporization of moisture, either in the coal or on surrounding surfaces, may require an additional amount of energy. If these losses are large enough, all the heat may be dissipated before the ignition temperature of the coal dust is reached.

PROPAGATION OF A COAL-DUST EXPLOSION

By definition, the initiatory phase of a coal-dust explosion ends when the source of ignition ceases to supply energy; after this the explosion must be self-sustaining or become extinguished. The question is resolved into a balance between production of energy by combustion of coal dust and its use in maintaining the explosion plus dissipation through losses.

Production of Energy

The rate of production of energy in the form of heat depends entirely on the rate of combustion of the coal dust, and this will be influenced by (1) the initial rate established by the source of ignition, (2) the density of the coal-dust cloud, (3) the size of the coal-dust particles, (4) the uniformity of their dispersion, (5) their reactivity with oxygen, particularly as influenced by volatile-matter content, and (6) the presence in the cloud of nonreactive material such as inherent ash and moisture of the dust and admixed incombustible dust and water. In commercial mines any of these may vary widely in different instances. In large-scale testing it is desirable to keep all of them, except the one under investigation, as nearly constant as possible; but there are limitations to the possibility of doing this.

6/ For a description of these sources of ignition and their properties, see Rice, George S., Greenwald, H. P., and Howarth, H. C., Explosion Tests of Pittsburgh Coal Dust in the Experimental Mine, 1925 to 1932, Inclusive. U. S. Bureau of Mines Bulletin 369, 1933, pp. 13-25.

Energy Consumed by the Explosion

If the explosion is to continue, energy must be consumed in three ways - (1) in formation of the dust cloud in advance of the flame, (2) in heating the coal dust to its temperature of ignition, and (3) in maintaining pressure in the zone behind the flame. The explosion will be extinguished if any one of these is absent. Combustion of the coal dust produces energy only in the form of heat, and for formation of a dust cloud the heat must be transformed to mechanical energy. This occurs through the agency of either pressure waves or bodily movement of masses of gas. Heat energy is used directly in bringing the coal dust to its ignition temperature. Maintenance of pressure behind the flame may result from the individual or concurrent action of residual heat and compression of the mass of gas present after combustion into less volume than it would occupy at normal temperature and pressure.

Loss of Energy

Losses accompany consumption of energy in the three ways mentioned above. The mechanical energy of pressure waves and gas movements is dissipated by friction of gases on walls and obstructions, is transferred to solid objects that are moved, and is lost by expansion at side openings. Heat is lost directly by transfer to the surrounding walls and other solid objects and by heating inert dust or coal dust in excess of that which can be burned by the available oxygen. Compressed gases behind the flame suffer losses by direct transfer of heat to the surroundings, by expansion under some conditions, and by mechanical action if the gases are moving. At times these losses may carry pressure below that of the atmosphere, and this may act to retard advance of the flame of the explosion.

SELECTION OF CONDITIONS FOR LARGE-SCALE TESTS

The investigator conducting large-scale tests of the explosibility of coal dust will wish to have his data directly applicable to conditions in commercial mines. From the foregoing it is evident that he must base his test methods on the conditions in those mines. As it is impractical if not impossible to attack the problem from the fundamental standpoint of production and dissipation of energy, he must concern himself instead with the secondary phenomena arising from the energy changes. This makes it necessary to reproduce in the test method the original mine conditions with as few alterations as may be possible. To test all the conditions in commercial mines under which coal-dust explosions may be initiated and propagated is an impossible and also an unnecessary task. Work may be concentrated on those conditions that are most dangerous and also are most likely to occur.

Experiments have shown that the factors influencing the energy changes may be grouped under seven heads: (1) Reactivity, particularly as related to the volatile ratio of the coal; (2) size (or specific surface) of the dust; (3) quantity of coal dust and inert dust present; (4) original location of the respective dusts in the cross section of the testing passageway; (5) inflammable gas in the air; (6) dust-raising and igniting power of the source of

ignition; and (7) cross sectional area and shape of the testing passageway, its length, the presence of bends, obstructions, and openings of all kinds, and the character and condition of the walls, such as smooth or rough and wet or dry. It is desirable that the investigator be able to make changes in all of these at will, and difficulty usually will be experienced only with the seventh. The more nearly the conditions of the testing passageway approach those of a coal mine, the less uncertainty will there be in application of the test data. The United States Bureau of Mines published a summary of its findings under the above seven heads in 1929 in Technical Paper 464.[1] Additional work with Pittsburgh coal dust was set forth in Bulletin 369, to which reference has been made. There remain unpublished the data of tests of a number of different coals made since 1929.

After deciding under what conditions his tests shall be made, the investigator has choice of two general methods of procedure. He may view the problem from an abstract standpoint and determine the effect of each of the seven factors mentioned above within the limits between which they vary in commercial mines. The goal of such an investigation would be a mathematical expression combining all of them in such form that known values could be substituted after study of a commercial mine and the rock-dusting requirements of the mine or any portion thereof could be evaluated without further experiment. Attempts to do this have been made, but none has been completely successful so far because not all the factors were included. The authors believe that development of such an expression or mathematical formula is a desirable goal but that it can be accomplished only by far more extensive testing than has been done as yet.

It follows that until such a formula is available, large-scale tests of the explosibility of coal dust should be divided into groups according to the factor on which they give information, and due care should be taken to prevent inferences that are not warranted by the test conditions.

APPLICATION OF TEST DATA

Obviously, application of test data to a specific mine can be little more than a guess unless conditions in the mine in question are known, and this knowledge should embrace all of the seven factors detailed above. Recognition of this fact led to early development in the Bureau of Mines of methods of investigating coal-dust explosion hazards in both gassy and nongassy commercial mines. Particular attention is paid to possible sources of ignition and the conditions under which they may be active. The quantity and distribution of the dust are noted. Numerous dust samples are taken in working places and passageways so that particle-size tests and analyses in the laboratory give a reasonably satisfactory picture of these factors. The coal is sampled in place in the bed by standard methods to obtain the range of composition in different parts of the mine. Air samples are taken in all ventilating splits and in the main return to determine what allowance must be made for inflammable gas in estimating rock-dusting requirements. Then a quantity of specially mined coal, sufficient for large-scale tests, is taken from the coal bed under consideration and sent to the Experimental mine. The foregoing information is

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[1] Rice, George S., and Greenwald, H. P., Coal-Dust Explosibility Factors Indicated by Experimental Mine Investigations, 1911 to 1929: Tech. Paper 464, Bureau of Mines, 1929, 45 pp.

used to determine which tests are more important as well as to make application of the test data. The authors' experience leads them to believe that this branch of the work can scarcely receive too much attention and that it furnishes the only reliable method of making application of test data.

The question of probability of occurrence of a coal-dust explosion in the specific mine must be considered. For example, a mine may be extremely gassy, and there is a possibility that a large body of an explosive gas-air mixture may be ignited. On the other hand, the system of ventilation employed, the use of approved explosives, apparatus, and machinery, and the vigilance of the management in maintaining safe conditions may make an explosion quite improbable. Security against the propagation of a coal-dust explosion under any condition that can possibly occur in a mine having a very large inflow of gas might call for maintenance by rock-dusting of 75 or 80 percent inert matter in the mine dust; this would include inert matter naturally present. The judgment of the trained investigator enters here, as it will at other points, to provide adequate precautionary measures, and the authors believe that it will never be wholly supplanted.

COMPARISON OF TEST DATA OBTAINED BY DIFFERENT INVESTIGATORS

Different investigators have selected different test methods, each according to his particular needs or viewpoint. It is inevitable that one will obtain results differing from those of another who uses different methods. Discussions involving comparison of test data obtained in different countries are fruitless unless the differences in test methods are taken into account, and at present there is no entirely satisfactory way of evaluating these.

The authors suggest the desirability of all investigators adopting some standard method of test by which their work can be compared. It is not supposed that this method would supplant those now used by any investigator; the purpose is to obtain supplemental evidence, through use of a test made under an agreed standard, to provide a satisfactory and logical basis for the common evaluation of all large-scale tests.

Such a method assumes use by all investigators of a standard piece of testing equipment, and this must of necessity be relatively inexpensive. The authors suggest that, to begin with, it might well be on the laboratory scale.

Laboratory tests may have two purposes - (1) empiric reproduction of large-scale tests for routine purposes to save time or costs and (2) research work on such factors as are amenable to laboratory treatment. Fulfillment of the first purpose has been achieved in England and the United States through use of a device described below. With regard to the second purpose, laboratory testing can deal only with factors affecting the initiation of a coal-dust explosion, such as are enumerated on pages 2 to 3. The work that can be done on dust-cloud formation is evidently limited, and ignition of a preformed cloud will be a more fruitful field of work. It should be possible to determine relation of ignitibility to composition or reactivity, density of the dust cloud,

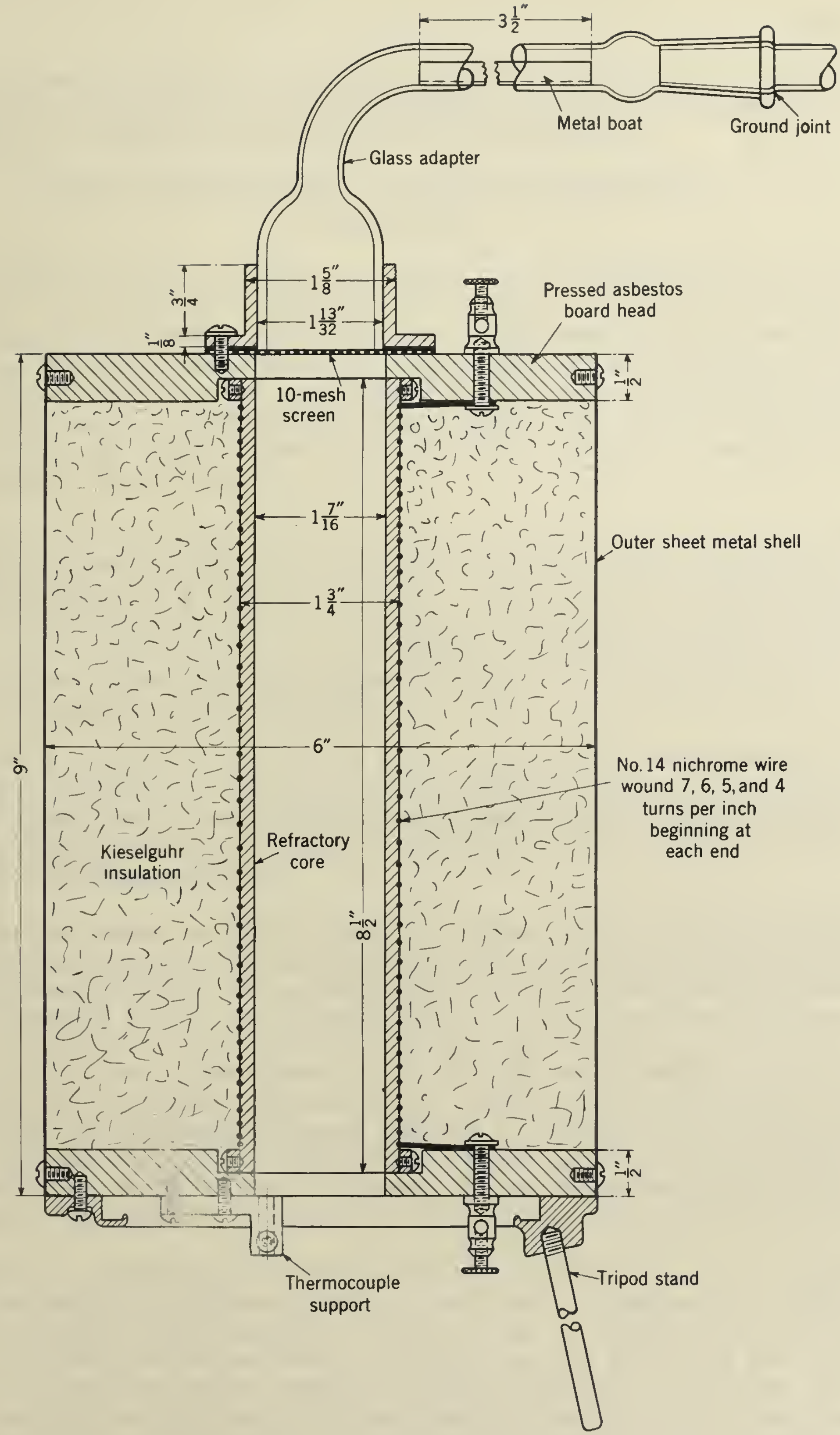

Figure 1.—Cross section of laboratory furnace for determining inflammability of small samples of dust.

size of particles (within limits), and temperature and nature of the igniting source, either individually or in combination. The effect of added inert dust in preventing or controlling ignition can be studied also.

PROPOSED STANDARD METHOD OF TESTING INFLAMMABILITY OF COAL DUSTS

All who have pursued laboratory studies of the inflammability of coal dusts know of the method by which the dust to be tested is blown through a heated furnace. This method has been used in different forms by many investigators in the past, has been developed by the British investigators to meet their needs, and with some modifications has been adopted by the United States Bureau of Mines as its standard laboratory method. The apparatus used either in England or in the United States might possibly serve as a satisfactory standard for all other countries.

Figure 1 is a cross sectional view of the laboratory furnace developed at Sheffield, England, by Dr. A. L. Godbert, of the staff of the Safety in Mines Research Board, and further tested by him at the Pittsburgh Experiment Station of the Bureau of Mines. The furnace is cylindrical, with dimensions as shown. In the original English development, oxygen was used to blow a cloud of the dust under test through the furnace; air was substituted for oxygen in the American development.[8]

The dust was placed in the metal boat shown in the glass adapter (fig. 1). More recently, Godbert, at Sheffield, has abandoned the boat and placed the dust in the glass tube to the right of the ground joint. At Pittsburgh, a metal dust container has been placed permanently in the apparatus at the location through a removable cover.

By use of a proper temperature in the furnace (720° C.) and adjustment of the volume and pressure of the air used to carry the dust into the furnace, it has been found possible to reproduce results of standard propagation tests in the Experimental mine. This development is for routine test purposes.

Use of this piece of equipment in a standard manner by all countries will leave as possible variables only the first three of the seven groups of factors mentioned above as influencing energy changes in coal-dust explosions. It would be a decided advance to have all investigators come to an agreement on these three, and a study of test methods by which the others might be attacked could follow. For example, comparative data through laboratory testing have

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8/ A description of the English work was given by Godbert, A. L., and Wheeler, R. V., The Relative Inflammability of Coal Dusts; A Laboratory Study: Safety in Mines Research Board Paper 56, 1929, 23 pp. A simpler form, for purely routine purposes, was described by Godbert, A. L., A Routine Test of the Inflammability of Mine Dusts: Safety in Mines Research Board Paper 68, 1931, 9 pp. The work at Pittsburgh is described by Godbert, A. L., and Greenwald, H. P., Laboratory Studies of the Inflammability of Coal Dusts; Effect of Fineness of Coal and Inert Dusts on the Inflammability of Coal Dusts: Bull. 389, Bureau of Mines, 1936, 29 pp.

been obtained on the relative inflammability of different American and British coals by this method, and the test data are included in Bureau of Mines Bulletin 389. These data enable comparisons to be made with large-scale test results obtained on the same coals at the respective stations. It is suggested that for mutual benefit each investigator should parallel any series of large-scale tests made at his respective station with tests in the laboratory, unless the variations whose influence is being studied on the large scale can not be reproduced in the laboratory tests. The laboratory work should include a series of special tests made under an agreed method that might be revised from time to time by mutual consent.

SUMMARY AND RECOMMENDATION

The authors' experiences with and beliefs concerning large-scale tests of the explosibility of coal dust may be summarized as follows:

1. The continuance of a coal-dust explosion depends on production of sufficient heat by combustion of coal dust to provide the thermal and mechanical energy requisite for the process and at the same time provide for the loss of energy to surroundings that is inevitable.

2. It is not possible to study coal-dust explosions from the standpoint of production, consumption, and loss of energy, and secondary phenomena must be observed.

3. Conditions governing energy changes in coal-dust explosions vary widely in commercial mines, and the investigator should have thorough knowledge of these before fixing the conditions of his large-scale tests.

4. The data obtained by different investigators have been inconsistent at times because test methods were not comparable and knowledge by which they could be harmonized has not been available.

5. The results obtained by different investigators can be correlated if all will adopt and use (for reference purposes only) some simple test method and equipment.

6. A start can be made by use of the laboratory furnace method of testing dusts; this already has been done by American and British investigators. By means of it, an agreement should be reached on the influence of size and composition or reactivity of coal dust.

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